622.09
Un 32 mo
cop. 2

R. I. 3300

3-28-36

FEBRUARY 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

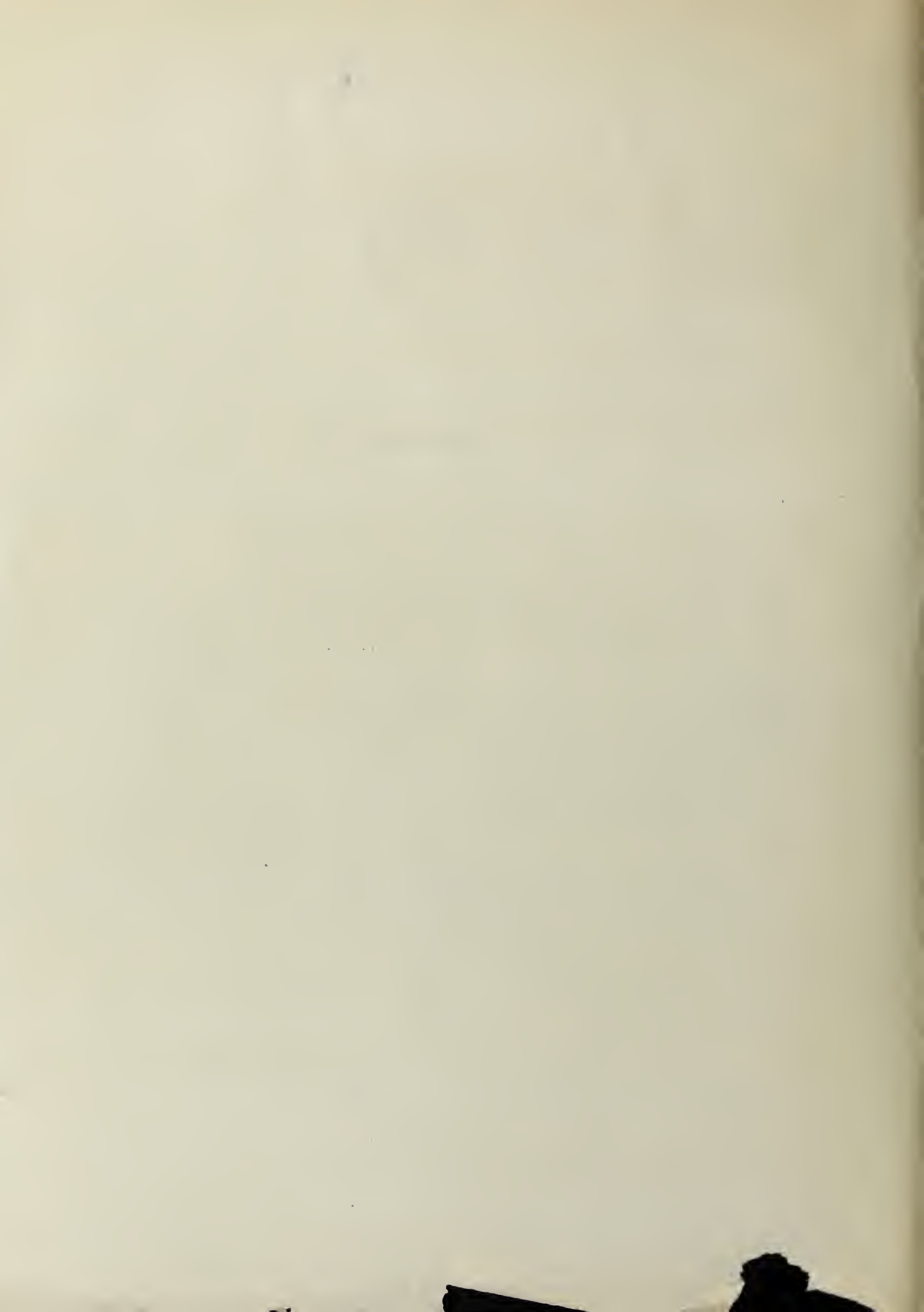
FLOTATION OF LANGBEINITE FROM THE POTASH FIELD OF NEW MEXICO AND TEXAS

THE LIBRARY OF THE
JUL 23 1936
UNIVERSITY OF ILLINOIS

BY

F. D. DEVANEY AND S. R. B. COOKE

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.



R. I. 3300
February 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

FLOTATION OF LANGBEINITE FROM THE POTASH FIELD OF NEW MEXICO AND TEXAS[1]

By

F. D. DeVaney[2] and S. R. B. Cooke[3]

INTRODUCTION

The potash mineral receiving the most attention in the newly discovered New Mexico-Texas potash district near Carlsbad, N. M., is sylvite (potassium chloride). Sylvite contains more potassium than any other mineral. Merchantable high-grade deposits have been found, and those of lower grade may be enriched to a high degree of purity by refining methods. Deposits containing sylvite were the first to be developed and are the only ones being mined.

However, many other minerals that contain potash occur in the salt beds of the Carlsbad district. Of these the more important are polyhalite, carnallite, langbeinite, leonite, and kainite. The nature of the deposits and their mineralogy have been described in detail by members of the United States Geological Survey.[4]

Langbeinite, although not so abundant in these deposits as carnallite and sylvite, has particular interest because the potash is in the form of the sulphate and as such commands a premium in terms of K_2O units. This is explained by Fox and Turrentine[5], who state:

1/ This paper presents the results of work done under a cooperative agreement between the Bureau of Mines and the Missouri School of Mines and Metallurgy, Rolla, Mo. The Bureau of Mines will welcome reprinting of this report, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3300."

2/ Associate metallurgist, U. S. Bureau of Mines, Rolla, Mo.

3/ Research metallurgist, Missouri School of Mines and Metallurgy, Rolla, Mo.

4/ Schaller, W. T., and Henderson, E. P., Mineralogy of Drill Cores from the Potash Field of New Mexico and Texas: U. S. Geol. Survey Bull. 833, 1932, 124 pp. Smith, H. I., Potash Development in Southeastern New Mexico; Contr. 52, Am. Inst. Min. and Met. Eng., 1933.

5/ Fox, E. J., and Turrentine, J. W., Potassium Sulphate: Ind. and Eng. Chem., vol. 26, 1934, p. 493.

The basis is the demand that fertilizers designed for certain crops, notably tobacco, shall be largely chlorine-free. It is this fact rather than the plant-food value of sulphur or the less objectionable properties of the sulphate as compared with the chloride radical that assign a value of $42 per ton for 90 percent potassium sulphate, as compared with $34 per ton for the 80 percent potassium chloride, or on a potassium oxide basis, of $87 per ton as sulphate, compared with $67 per ton for the chloride.

Smith[6/] says, "It (langbeinite) is an excellent fertilizer and can readily be converted to the sulphate of potash for export trade."

Drill records show the presence of beds of langbeinite that are thick enough to lead to the belief that large amounts are available. Much of the langbeinite is low-grade, and if the deposits are to be utilized a method of concentration must be developed.

Inasmuch as shafts have not been sunk to the langbeinite beds drill cores afforded the only material available for concentration studies. The sample came from Government test well 17 in sec. 27, T. 20 S., R. 33 E., Lea County, N.Mex., and was supplied by the United States Geological Survey. The horizon from which the cores came was between 2,514 and 2,560 feet. The material sent to Rolla was not a true sample of the bed, as the best of the specimens containing langbeinite had been reserved for museum use. The low-grade material remaining constituted the ore on which concentration tests were made and all tests were of necessity made with small lots.

MINERALOGY

Langbeinite is a double sulphate of potassium and magnesium having the composition $K_2SO_4 \cdot 2MgSO_4$. The pure mineral contains 22.7 percent K_2O, 19.5 percent MgO, and 57.8 percent SO_3. In the sample examined the langbeinite occurred as small crystals ranging in color from white to faint pink. The mineral has a specific gravity of 2.858. Its hardness, 3 to 4, is high for a potash mineral. It may be distinguished from halite, with which it is usually associated, by the fact that it is tasteless and has a conchoidal or irregular fracture with no cleavage. Langbeinite effloresces slightly, and cores exposed to the air show a white efflorescence due to the alteration of langbeinite to some of the hydrous magnesium sulphates. As langbeinite is much less soluble in cold water than halite or sylvite some idea of the amount of langbeinite present may be gained by a quick wash; the chlorides dissolve rapidly and leave practically all the langbeinite as residue. Langbeinite occurs in granular form but more commonly as tetrahedrons which sometimes show characteristic triangular outlines on the core surface. Optically it is isotropic and has an index of refraction of 1.533.

The cores examined consisted essentially of two minerals, langbeinite and halite, in the proportion by weight of approximately 1 part of langbeinite to

---

6/ See footnote 4.

4 of halite. The analysis of the sample used was 4.75 percent K_2O. The halite crystals were much larger than the langbeinite.

CONCENTRATION

The methods used in concentrating the sylvite ore from this district 1/ suggested that langbeinite might be concentrated. Both the microscope and heavy liquids were used in the study of mineral association. These methods indicated that much finer grinding would be necessary to liberate langbeinite from halite than was required for the sylvite. Float-and-sink tests showed that only a small percentage of langbeinite is free at sizes coarser than 10-mesh. The results of these tests are given in table 1. The tests and microscopic examination indicated that if good recoveries of high-grade concentrates are to be made by physical separation grinding to 65-mesh is essential.

TABLE 1.- Float-and-sink tests of langbeinite

Size, mesh	Product	Weight, percent	Assay, percent K_2O	Percent of total K_2O
-6 + 8	Float on 2.75 sp. gr.	92.4	4.79	74.9
	Sink in 2.75	7.6	19.48	25.1
	Composite	100.0	5.90	100.0
-8 + 10	Float on 2.75	90.7	3.30	62.2
	Sink in 2.75	9.3	19.58	37.8
	Composite	100.0	4.81	100.0
-65 + 150	Float on 2.75			
	Sink in 2.75 sp.gr.	81.9	1/ 0.5	9.4
	Composite	18.1	1/22.0	90.6
		100.0	4.4	100.0

1/ Microscopic estimate.

The specific gravity of langbeinite is 2.86, whereas that of halite is 2.15. Inasmuch as the specific gravity of a brine saturated with respect to halite and the components of langbeinite is 1.30 at room temperature, the density of langbeinite relative to halite is 1 to 1.83. Such a range is more than ample to permit separation by tables if tabling were otherwise feasible. However, because of the necessity of grinding to 65-mesh, tabling is not promising. Selective agglomeration, too, as a possible concentration method does not appear promising owing to the fineness to which the ore must be crushed for liberation.

1/ Coghill, W. H., DeVaney, F. D., Clemmer, J. B., and Cooke, S. R. B., Concentration of the Potash Ores of Carlsbad, N. Mex., by Ore-Dressing Methods; Rept. of Investigations 3271, Bureau of Mines, February 1935, 13 pp.

When the investigation was begun it was hoped that a flotation process employing the same type of collectors as in the separation of sylvite from halite would be applicable for the separation of langbeinite from halite. This hope was fulfilled; by using a sulphated aliphatic alcohol it was found that the langbeinite could be floated away from the halite satisfactorily.

In the laboratory tests an ore crushed to 100-mesh was conditioned for a short time in a thick pulp with the reagent, Avirol 80. Brine saturated with respect to the ore - that is, as to sodium chloride, potassium sulphate, and magnesium sulphate - was used as the flotation medium. The specific gravity of this brine at 25°C. was 1.30. After the ore was conditioned brine was added to give a pulp consistency of 25 percent solids.

Rougher concentrates and finished tailings were made in the first flotation cell. The rougher concentrates were recleaned in a second cell, giving finished concentrates and middlings. No additional reagents were added in the cleaner cell. As much brine as possible was removed from the concentrates, middlings, and tailings immediately after flotation by filtration and centrifuging. The results of one of the limited number of flotation tests are given in table 2.

TABLE 2.- Flotation test of langbeinite ore

Product	Weight, percent	Assay, percent K_2O	Calculated as langbeinite, percent	Percent of total K_2O
Cleaner concentrates	14.2	21.68	95.5	72.6
Middlings	8.8	5.51	24.2	11.4
Rougher tailings	77.0	.88	3.9	16.0
Composite	100.0	4.24	18.7	100.0

High-grade concentrates approaching the theoretical analysis of langbeinite were made with a fair recovery. The middlings consisted of unlocked langbeinite and halite and undoubtedly could be re-treated to recover more langbeinite. The fact that the tailings in the tests are somewhat higher than desirable does not necessarily mean that they cannot be improved.

Just how much reagent would be required in practice where the brine would be recirculated is not known. The amounts used are believed to be small and comparable with those required in the laboratory flotation of sylvite from halite.

Flotation of the langbeinite is rapid and positive, not over 5 minutes being required to complete a batch test. The separation is not unduly sensitive to small changes in amount of reagent, pulp consistency, or temperature.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

R. I. 3302 May 1936

DEPARTMENT OF THE INTERIOR
UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

MICROCOLORIMETRIC DETERMINATION OF BENZENE IN BLOOD AND URINE

BY

S. J. PEARCE, H. H. SCHRENK, AND W. P. YANT

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

622.09
Bu3Rino
no.3302-3315
cop.2

R. I. 3302
May 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

MICROCOLORIMETRIC DETERMINATION OF BENZENE IN BLOOD AND URINE[1]

By S. J. Pearce,[2] H. H. Schrenk,[3] and W. P. Yant[4]

INTRODUCTION

The need for a simple, very sensitive, and accurate method for the determination of benzene in blood and urine was evident throughout a recent study of chronic benzene (C_6H_6) poisoning conducted cooperatively by the United States Bureau of Mines, The Barrett Company, and the producers of benzene. A combustion method, which will be described in a later publication, and a previously described[5] adaptation of Smyth's method were developed and used during that investigation. The adaptation of Smyth's method gave satisfactory results when dealing with exposure conditions that produced chronic poisoning in a comparatively few days, but it was time-consuming and lacked the accuracy necessary for a study of the absorption, distribution, and fate of benzene in the body during exposure to threshold concentrations of benzene vapor in air. The combustion procedure was satisfactory from the standpoint of sensitivity but was cumbersome, intricate, time-consuming, and required the use of special and somewhat expensive physical and chemical apparatus. Following the termination of this study of chronic poisoning, possibilities for developing better methods occurred, and these have been investigated by the Bureau of Mines with the aid of Relief Works Division personnel. A microcolorimetric method for the determination of benzene in air has been described recently.[6] The present report deals with the adaptation of this new colorimetric method to the determination

HARDY
DEC 17 1951

1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3302."

2/ Assistant chemist, Gas Section, Pittsburgh Experiment Station, U.S. Bureau of Mines, Pittsburgh, Pa.

3/ Chemist, Gas Section, Pittsburgh Experiment Station, U.S. Bureau of Mines, Pittsburgh, Pa.

4/ Supervising chemist, Gas Section, and supervising engineer, Pittsburgh Experiment Station, U.S. Bureau of Mines, Pittsburgh, Pa.

5/ Yant, W. P., Schrenk, H. H., and Mautz, P. H., A Procedure for the Removal and Determination of Small Amounts of Benzene in Biological Material: Rept. of Investigations 3282, Bureau of Mines, 1935, 7 pp.

6/ Schrenk, H. H., Pearce, S. J., and Yant, W. P., A Microcolorimetric Method for the Determination of Benzene: Rept. of Investigations 3287, Bureau of Mines, 1935, 11 pp.

of benzene in blood and urine and suggests its use in the determination of benzene in other materials. Because of its simplicity and greater sensitivity, it has many obvious advantages over the other methods used by the authors for studying chronic benzene poisoning. It also possesses possibilities of use for making direct determination of benzene in the blood and urine in the control of exposure to benzene in industry.

BUREAU OF MINES METHOD

The benzene is removed by aeration from the sample of blood or urine contained in a bubbler and is swept by a current of air directly into a mixture of fuming nitric and sulphuric acid contained in another bubbler, where it is nitrated to m-dinitrobenzene. Following this, the procedure is the same as that described for the determination of benzene in air.1/ Briefly, the acid mixture is neutralized, extracted with one 10-ml portion of butanone, and 1.5 ml of 40-percent sodium-hydroxide solution is added to the butanone extract. A reddish-purple color develops, the intensity of which is proportional to the amount of m-dinitrobenzene present. A series of color standards containing known amounts of m-dinitrobenzene is prepared simultaneously for comparison.

APPARATUS

Aeration and nitration bubblers. - Figure 1 shows the types of bubblers used by the Bureau of Mines for aeration and nitration of benzene from blood or urine samples. Other similar types would be suitable. The blood or urine to be aerated is placed in bubbler a, connected to bead tower or bubbler b, which, in use, contains the nitration acid mixture. A calibrated aspirator bottle is connected at c for forcing benzene-free air through the bubblers. As up to 15 liters of air are required in some cases for complete aeration, the air may be supplied continuously from a bottle of that or greater capacity or by successive deliveries from a smaller bottle. Other means for providing benzene-free air at a roughly known rate would be suitable. The remaining apparatus is standard chemical laboratory equipment, as indicated in a previously published description of a procedure for the determination of benzene.1/

PROCEDURE

Place 5 to 10 ml of the blood or urine in bubbler a and add a few crystals of citric acid to urine and 0.5 ml of a mixture of 1.3 grams of calcium stearate triturated with 20 ml of white Russian mineral oil to blood to prevent foaming during aeration. (Caprylic alcohol was unsatisfactory because it produced a yellow color in the butanone extract, which changed the quality of the final color.) Place 2 ml of a mixture of equal parts by volume of concentrated sulphuric acid (sp. gr. 1.84) and fuming nitric acid (sp. gr. 1.50) in bubbler b. Connect bubblers a and b, using a film of water as a lubricant; also connect the aspirator bottle to c by a hose closed with a screw clamp. Adjust the aspirator arrangement so that there is positive pressure in the delivery bottle and open screw clamp carefully. For urine samples, the air rate should be adjusted to give a flow of 3 to 6 liters per hour, and

1/ Work cited. See footnote 6.

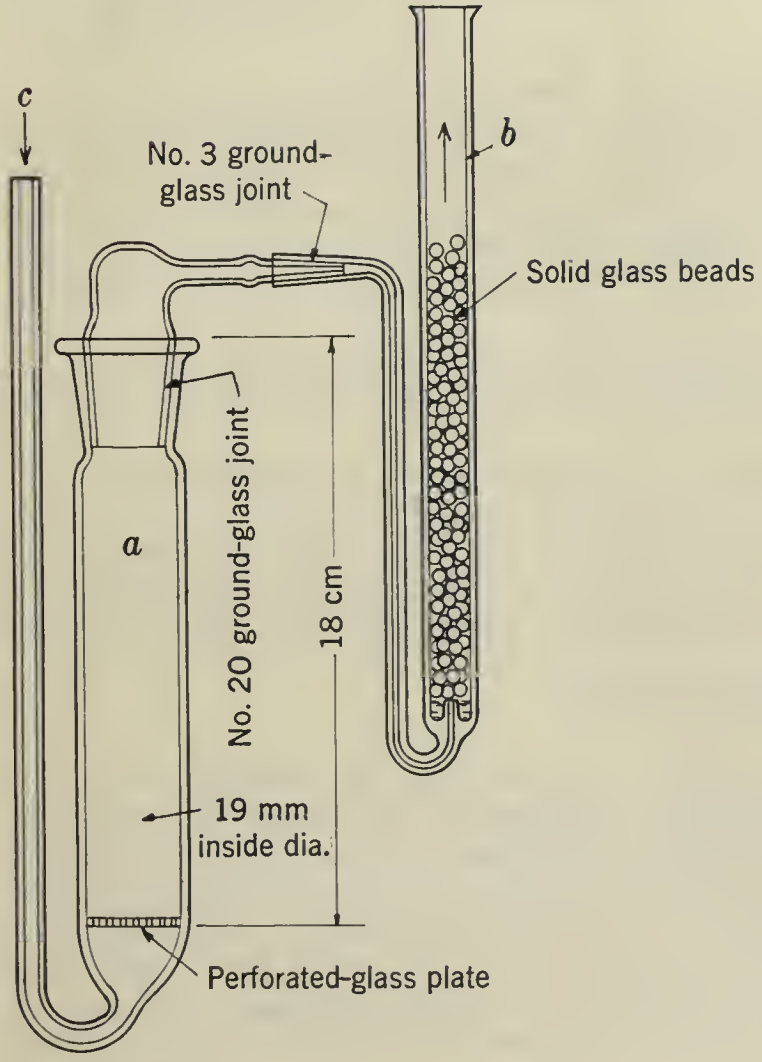


Figure 1.— Apparatus used for aeration of benzene from blood or urine and its nitration.

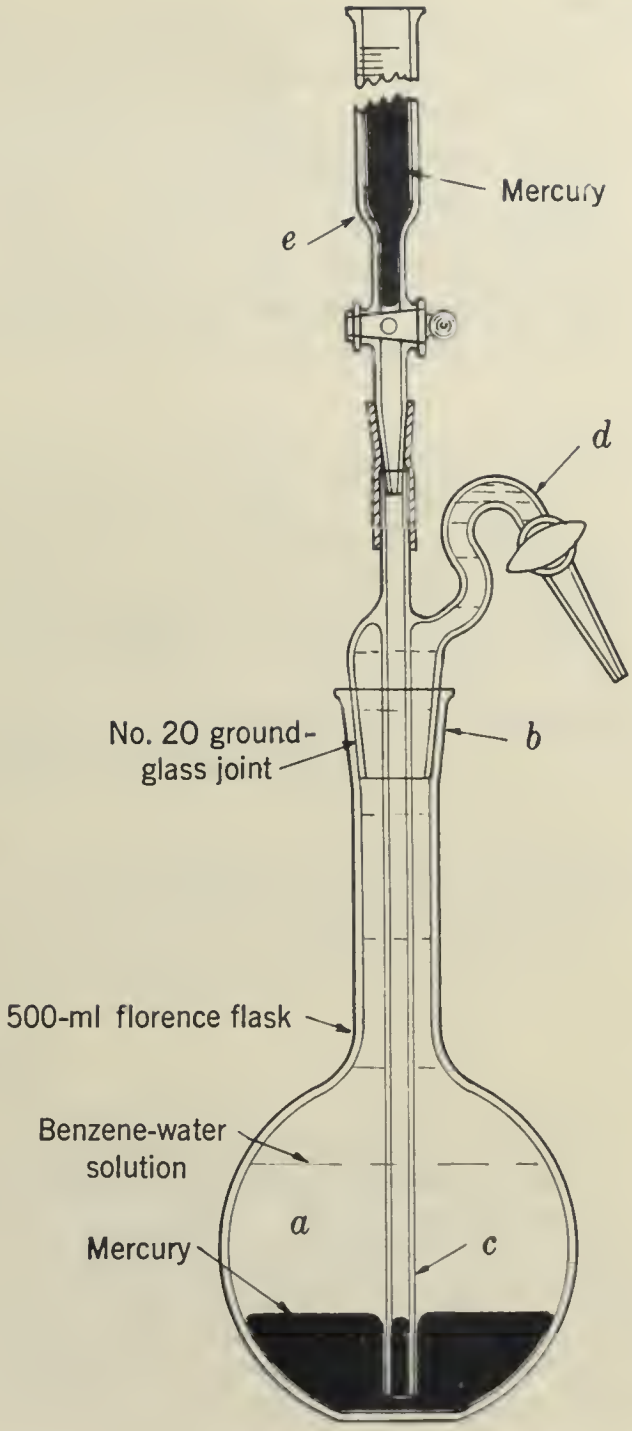


Figure 2.— Apparatus used for preparation of benzene-water solutions.

the flow should continue until approximately 6 liters of air have been passed through the sample. Benzene is removed with more difficulty from blood, particularly after addition of the oil-base antifoaming reagent. For amounts of benzene of the order of 0.1 to 0.3 mg in 100 ml of blood, and using a 5- to 10-ml specimen, aeration with 6 to 10 liters is sufficient. More aeration is required for higher benzene concentrations. If small bubbles are formed and good agitation produced the rate of flow of the air in the aeration procedure is not very important. From the standpoint of time saving in the use of large volumes the rate is governed primarily by the maximum capacity of the apparatus used. The maximum rate for the one shown in figure 1 is about 8 liters per hour.

Although 80 percent or more of the benzene is removed by about one-half the aeration mentioned in the foregoing, the extra treatment is required for greater accuracy. To check the completeness of the aeration, it is good practice to replace the original bubbler with one containing fresh nitration acid and continue aeration for an hour. The contents of this tube should be treated separately to ascertain if the benzene removal has been complete.

After aeration disconnect nitration bubbler b, and proceed as described in the colorimetric method for the determination of benzene.8/

Experience with this method has shown that benzene is aerated slowly from fat- or oil-containing materials. For the treatment of these materials, the apparatus described by Yant et. al.9/ for aeration accompanied by heating is recommended. A nitration bubbler of type b, figure 1, can be used if care is taken to prevent back-aspiration.

TESTS OF METHOD

The sensitivity and accuracy of the method were investigated by using specimens of blood and urine that contained known amounts of benzene and subjecting them to the prescribed analytical procedure. These specimens were prepared by adding measured amounts of a standard aqueous benzene solution to human urine and beef blood. The urine specimens were obtained fresh each day. The blood was treated with 2 grams of potassium oxalate per liter to prevent coagulation and 0.6 gram of bichloride of mercury per liter as a preservative. The blood was kept on ice and was used over a period of about 2 months.

Device and Procedure for Preparing Standard Benzene-Water Solution

In a previous investigation9/, standard benzene-Nujol solutions were used because it was found that, owing to the comparatively low solubility of benzene in water, considerable benzene was lost to the air space created

8/ Work cited. See footnote 6.
9/ Work cited. See footnote 5.

above the water in a volumetric flask by removal of successive portions of standard solution. A benzene-oil standard was not desired for the present work. The loss of benzene from the aqueous standard was prevented by preparing it in an apparatus from which it was removed by mercury displacement, thereby eliminating the air space above the standard.

Figure 2 shows the apparatus used for preparing standard benzene-water solutions. The apparatus consists of a 500-ml Florence flask, a, which is closed by a ground-glass stopper, b. The stopper is fitted with an inlet tube, c, which extends to within about 4 mm of the bottom of the flask, and an outlet tube, d, to which is attached a stopcock and burette tip. The tip of a burette, e, for admitting mercury is fitted into the top of the inlet tube and the joint made tight by a collar of rubber tubing. The stopcocks are carefully ground, and a film of water is used as a lubricant. It is impractical to use ordinary stopcock grease owing to the solubility of benzene in it.

Calibrate the flask to a point midway on the neck. Add distilled water to fill the flask to the calibration mark. Close the outlet and inlet openings of the flask, the latter by means of rubber tubing and a screw clamp or preferably by a stub of glass rod with the end butting against the glass inlet tube, to prevent contact of the benzene-water solution with the rubber during the shaking period.

The benzene is introduced by placing it in a sealed glass bulb made by drawing out a piece of clean glass tubing to a thin-walled capillary about 1 mm in diameter, melting-in the end, and blowing a thin-walled bulb about 7 mm in diameter. The capillary tubing, about 3 cm long, is bent in the middle to form an angle of about 45°. It is then hung over the edge of a test tube completely filled with benzene. The bulb is alternately heated and cooled until the estimated required amount of benzene is drawn into the bulb. Any benzene in the capillary is removed. The amount of benzene is judged from experience and must be a quantity less than that required for saturating the volume of water used at room temperature. The amounts used ranged from 5 to 45 mg. The bulb is placed in a depression in a small block of asbestos and is protected from the heat of a needle-point gas flame by allowing the capillary to protrude through a sheet of aluminum foil which covers the bulb. The capillary is sealed off between the bend and the bulb, as close to the bulb as possible. Both the cleaned bulb and sealed-off piece of capillary are weighed carefully.

Place the bulb on top of the water and push to the bottom of the flask by the inlet tube, which crushes the bulb as the stopper is firmly seated. Shake the flask vigorously for 10 minutes, inverting several times to allow water to run into the inlet tube and dissolve any benzene contained therein. Tape the stopper securely in place. Immediately attach a 50-ml burette containing grease-free mercury to the inlet, as shown in figure 2; open the stopcock on the flask and run mercury slowly into the flask until the air above the solution is displaced completely. If care has been taken in the

preparation of this solution, the concentration of benzene will deviate but little from the calculated value. However, the concentration of benzene in the solution should be checked by aeration and nitration of a measured volume. (No appreciable change was noted in the concentration of a solution used over a period of about 2 months.) For measuring quantities greater than 1 ml, displace the solution from the flask through the outlet tube by adding a measured quantity of mercury from the burette - i.e., 1 ml of mercury displaces 1 ml of solution. For measuring quantities smaller than 1 ml, greater accuracy is attained by displacing the solution as above into a graduated 1-ml pipette, the top of which is held in contact with the tip of the exit tube, the bottom being closed by a finger. The required amount of solution is then quickly measured from the pipette. Add the desired amount of standard solution directly to the blood or urine in the aeration bubbler.

EXPERIMENTAL RESULTS

The ranges of concentration of benzene in the blood or urine samples used for investigation of the sensitivity and accuracy of the method were chosen to correspond approximately to the concentration ranges found in previous experimental work with chronic benzene poisoning. These ranged from 0.1 to 3.0 mg of benzene per 100 ml of blood, and 0.2 to 25 mg of benzene per 100 ml of urine.

Table 1 gives the results obtained by analysis of samples of urine to which known amounts of benzene had been added.

The results of the analyses shown in table 1 indicate that benzene can be aerated from urine at room temperature and that 4 to 6 liters of air is sufficient for the removal of the quantities studied, and also that nitration proceeds satisfactorily under these conditions. As the colorimetric method may be considered to be accurate to $\pm$ 5 percent with ordinary usage, and accurate to $\pm$ 2 percent when extreme care is used, the percent recovery apparently is as accurate as the method.

Table 2 gives the results obtained by analysis of samples of blood to which known amounts of benzene had been added.

TABLE 1. - Results of analysis of urine containing known amounts of benzene

Benzene-water solution		Urine, ml	Air flow		Benzene	
Ml used	Benzene, mg/ml		l/hr.	Hrs.	Calculated, mg	Recovered, mg
0.45	0.050	9.0	2.7	1.5	0.022	0.022
.80	.050	9.2	3.0	1.5	.040	.040
1.00	.050	9.0	2.7	1.5	.050	.050
1.00	.050	9.0	2.7	1.5	.050	.050
1.00	.050	9.0	2.7	1.5	.050	.050
2.00	.050	8.0	3.0	1.5	.100	.103
4.00	.050	6.0	4.0	1.5	.200	.198
6.00	.050	4.0	4.0	1.5	.300	.309
7.00	.050	2.0	4.0	1.5	.350	.361
1.00	.880	9.0	3.0	1.5	.880	.913
2.00	.794	8.0	3.7	1.5	1.588	1.680
3.00	.880	7.0	3.0	1.5	2.640	2.470
3.00	.794	7.0	3.7	1.5	2.382	2.380
3.00	.794	7.0	3.7	1.5	2.382	2.340

TABLE 2. - Results of analysis of blood containing known amounts of benzene

Benzene-water solution		Blood used, ml	Air-flow		Benzene	
Ml used	Benzene, mg/ml		l/hr.	Hrs.	Calculated, mg	Recovered, mg
1.00	0.013	10.0	2.6	2.0	0.013	0.012
1.00	.013	9.0	2.9	2.25	.013	.012
1.00	.013	9.0	3.3	2.0	.013	.012
1.00	.013	9.0	3.3	2.0	.013	.012
1.00	.013	9.0	3.3	2.0	.013	.013
2.50	.013	7.5	2.9	2.25	.032	.029
2.50	.013	7.5	2.9	2.25	.032	.029
2.50	.013	7.5	3.3	2.0	.032	.035
2.50	.013	7.5	3.3	2.0	.032	.030
2.50	.013	7.5	3.3	2.0	.032	.035
1.00	.093	9.0	7.4	2.0	.093	.092
1.00	.093	9.0	7.4	4.0	.093	.094
1.00	.093	9.0	7.4	6.0	.093	.095
3.00	.093	7.0	7.5	2.0	.280	.263
3.00	.093	7.0	7.5	2.0	.280	.280

The results of the analyses shown in table 2 indicate that benzene can be aerated from blood at room temperature. Although approximately 5 to 45 liters of air was used for the removal of the quantities studied, a volume of about 6 liters undoubtedly is sufficient in most cases for 0.01 to 0.03 mg in 7.5- to 10 ml-specimens. When the concentration of benzene is higher than 0.03 mg in 10 ml, more aeration is required. With 0.28 mg in 10 ml, aeration for 2 hours at a rate of 7.5 liters an hour was sufficient. In addition to the data given in table 2, a 10-ml blood specimen containing 0.28 mg benzene was aerated at the rate of 6 liters per hour. The following accumulative percent recovery of benzene was obtained: 15 minutes, 83.5 percent; 30 minutes, 98.6 percent; 60 minutes, 101.0 percent; and 120 minutes, 101.0 percent.

The difference in time required for aeration of blood and aeration of urine is undoubtedly due largely to the use of an oil-base antifoaming reagent for the blood determinations.

The results of analyses given in tables 1 and 2 were obtained when samples of blood and urine containing known amounts of benzene were placed directly in the bubblers and aerated immediately. In practice it is not always possible to do this, and the sample must be stored in a cool place under mineral oil to prevent loss of benzene. The results of analyses of a series of benzene-blood samples to which 5 ml of mineral oil was added indicated that aeration at room temperature was not practical under these conditions, as the mineral oil tended to hold back the benzene during aeration.

Two samples of blood containing 0.28 mg of benzene to which 5 ml of mineral oil was added were heated and aerated in an apparatus similar to that described by Yant et al[10/] for the removal of small amounts of benzene from biological material. The volume of the flask used was 200 ml and the length of the reflux condenser 30 cm. The nitration bubbler (b, fig. 1) was attached to the outlet tube of the condenser. The flask was heated in a paraffin bath at 120°C. for 2 hours, and benzene-free air was forced through the apparatus at a rate of approximately 6 liters per hour. To determine the rate of removal of the benzene from the samples, the nitration bubbler on each apparatus was replaced with others containing fresh portions of nitrating acid at the end of 15, 30, and 60 minutes of aeration. Analyses of these separate acid portions by the microcolorimetric method showed the following average accumulative percent recovery of benzene: 15 minutes, 92.3 percent; 30 minutes, 97.7 percent; 60 minutes, 100.0 percent; 120 minutes, 100.0 percent.

In the presence of considerable amounts of mineral oil or other fatty material, it is recommended that the sample be heated as described to assure complete removal of the benzene in a relatively short period of time.

10/ Work cited. See footnote 5.

SUMMARY

A microcolorimetric method for the determination of small amounts of benzene (C_6H_6) in blood or urine is described. The benzene is aerated from blood or urine and swept into a small amount of nitration acid, after which the method described in Bureau of Mines Report of Investigations 3287 is followed. The method is sensitive to at least 0.001 mg benzene and has an accuracy of about 10 percent of the amount present in the range of 0.01 to 0.9 mg in 10-ml specimens. Apparatus and procedures are described in detail, and experimental data are presented. The use of the method for determining benzene in other materials is apparent.

ACKNOWLEDGMENTS

The authors wish to acknowledge the assistance of C. L. Allard, assigned through the Allegheny County (Pa.) Relief Works Division, in making many of the determinations.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

622.09
Un32mo
Cop. 2

3303 - 3375
Swamp

622.09
Un32mo
COP. 2

R. I. 3303

MAY 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

THE LIBRARY OF THE
FEB 17 1937
UNIVERSITY OF ILLINOIS

REPORT OF INVESTIGATIONS

COMPARISON OF OUTPUT AND INTAKE CHARACTERISTICS OF NATURAL-GAS WELLS IN TEXAS PANHANDLE FIELD

BY

M. A. SCHELLHARDT AND E. L. RAWLINS

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3303
May 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

COMPARISON OF OUTPUT AND INTAKE CHARACTERISTICS OF NATURAL-GAS WELLS IN TEXAS PANHANDLE FIELD[1]

By M. A. Schellhardt[2] and E. L Rawlins[3]

CONTENTS

	Page
Introduction	2
Acknowledgments	2
Scope of report	3
Outflow and inflow tests on specific gas wells	3
Selection of test wells	4
Source of gas used for inflow tests	4
Pressure gages and metering equipment	4
Description of tests and data	5
Well A	5
Well B	11
Well C	12
Well D	14
Well E	17
Well F	19
Comparison of output and intake characteristics	19
Well A	19
Well B	20
Well C	20
Well D	21
Well E	22
Well F	22
Summary of results	22

1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3303."

2/ Associate natural-gas engineer, U.S. Bureau of Mines, Petroleum Experiment Station, Bartlesville, Okla.

3/ Senior petroleum engineer, U.S. Bureau of Mines, Petroleum Experiment Station, Bartlesville Okla.

INTRODUCTION

The Bureau of Mines has made periodic back-pressure tests on numerous gas wells in the Texas Panhandle field during the past several years to study the delivery capacities of the wells under different pressure and operating conditions and to determine the relationships between pressures and rates of delivery from gas wells.4/ However, information regarding the intake capacities of gas-well systems is meager, and in projects involving the flow of gas into porous-rock reservoirs it has been necessary to determine intake capacities of gas wells by trial. Accordingly the fundamental intake characteristics of natural-gas wells have been studied to develop a method for determining intake capacities under different wellhead pressures from data descriptive of the "output" or producing characteristic of the wells.

ACKNOWLEDGMENTS

The study of intake characteristics of gas wells in the Texas Panhandle field was made under the general supervision of R. A. Cattell, chief engineer, Petroleum and Natural-Gas Division, Bureau of Mines, Washington, D.C., and N. A. C. Smith, supervising engineer, Petroleum Experiment Station, Bureau of Mines, Bartlesville, Okla.

The work was carried on in cooperation with the Subcommittee on Gaging Gas-Well Deliveries, Main Technical and Research Committee, Natural-Gas Department, American Gas Association.

The authors are indebted to the officials, engineers, and field personnel of the Canadian River Gas Co., the Cities Service Gas Co., the Lone Star Gas Co., the North American Oil & Gas Co., and the Texoma Natural Gas Co. for their cooperation and assistance.

Special acknowledgment is due the following engineers: J. H. Dunn, Lone Star Gas Co., Dallas, Tex.; R. W. Camp, Consolidated Gas Service Co., Oklahoma City, Okla.; O. L. Carson, Panhandle Eastern Pipe Line Co., Amarillo, Tex.; J. R. Downes, United Gas Public Service Co., Houston, Tex.; C. J. Fowlston, North American Oil & Gas Co., Amarillo, Tex.; G. C. Kuhne, Texoma Natural Gas Co., Amarillo, Tex.; Max K. Watson, Canadian River Gas Co., Amarillo, Tex.; R. T. Wells, Cities Service Gas Co., Bartlesville, Okla.; L. M. Batton, Cities Service Gas Co., Pampa, Tex.; H. Clark, Cities Service Gas Co., Pampa, Tex.; G. B. Miller, Lone Star Gas Co., Shamrock, Tex.; and J. Probst, North American Oil & Gas Co., Skellytown, Tex.

4/ Pierce, H. R., and Rawlins, E. L., The Study of a Fundamental Basis for Controlling and Gaging Natural-Gas Wells. Part 1. - Computing the Pressure at the Sand in a Gas Well: Rept. of Investigations 2929, Bureau of Mines, 1929, 14 pp.; Part 2. - A Fundamental Relation for Gaging Gas-Well Capacities: Rept. of Investigations 2930, 1929, 21 pp. Rawlins, E. L., and Schellhardt, M. A., Back-Pressure Data on Natural-Gas Wells and Their Application to Production Practices: Monograph 7, Bureau of Mines, (in coop. with Am. Gas Asso.), 1936, 210 pp.

The authors are grateful to T. W. Johnson, natural-gas engineer, Bureau of Mines, Bartlesville, Okla. for assistance in conducting the field tests and preparing the report and to R. A. Cattell and Mr. Johnson for helpful criticism of the manuscript.

SCOPE OF REPORT

This report covers studies of the output and the intake characteristics of six natural-gas wells in the Texas Panhandle field. These data are supplemented by surveys of shut-in pressures of closed-in wells near three of the wells from which pressure and flow data were obtained.

OUTFLOW AND INFLOW TESTS ON SPECIFIC GAS WELLS

Investigations[5/] by the Bureau of Mines on the problem of gaging gas-well deliveries have disclosed a consistent relationship between rates of delivery of gas from a well and corresponding pressures when the pressure at the face of the sand in the well bore is used as the fundamental basis of interpretation. The relationship may be expressed mathematically by the formula

$$Q = C(P_f^2 - P_s^2)^n,$$

where Q = rate of flow, cubic feet of gas per 24 hours;
C = coefficient;
P_f = shut-in pressure at the sand face, pounds per square inch absolute;
P_s = back pressure at the sand face in the well bore while the well is flowing, pounds per square inch absolute; and
n = exponent.

The producing characteristics of natural-gas wells may be studied by plotting rates of gas delivery against the corresponding pressure factors $P_f^2 - P_s^2$ on logarithmic paper. For all practical purposes the relationship established by these plotted data is a straight line; for the purpose of this report the relationship between rates of gas delivery from a well and corresponding pressure factors $P_f^2 - P_s^2$ is termed the "output" or "outflow" characteristic of the well, and the test from which this relationship is obtained is termed an "outflow" test.

When gas is delivered into a gas well the pressure at the face of the sand must be higher than the pressure in the formation surrounding the well bore, and the rate of delivery into the well is determined by the difference of the squares of the pressure imposed at the face of the sand and the shut-in formation pressure ($P_s^2 - P_f^2$). The relationship between rates of gas delivery into a well and corresponding pressure factors $P_s^2 - P_f^2$ is here termed the "intake" or "inflow" characteristic of the well and the test from which this relationship is obtained, an "inflow" test.

5/ See footnote 4.

In conducting the outflow and inflow tests on the respective gas wells the wellhead pressures and corresponding rates of gas delivery were obtained as gas was delivered from or into each well. The outflow and inflow data were based on similar conditions of pressure and flow stabilization.

Selection of Test Wells

The wells on which inflow tests were made in this investigation were in Carson, Gray, Hutchinson, and Wheeler Counties, Tex. The selection of wells for studying the inflow characteristics was governed primarily by the following factors:

1. Distribution throughout the "sweet gas" area in the Texas Panhandle field (relatively free of hydrogen sulphide).

2. Condition of the well bore above the gas-producing strata with regard to cavings or liquids.

3. Availability of gas either directly from local wells or from compressor plants at pressures enough higher than the pressures on the respective intake wells to afford the necessary range of pressure-flow data.

Five of the wells selected probably were free from liquid or cavings, but the performance of the other well was affected materially by cavings or cuttings from the formation or by pipe scale from the gas-gathering system.

Source of Gas Used for Inflow Tests

The natural gas delivered to the underground reservoir through the respective test wells was obtained from other wells in the same general part of the field which were producing from the horizon to which the gas was returned. Gas delivered to the reservoir through four of the wells studied was transported directly from the source wells to the intake well through field gathering lines, and gas was delivered to the reservoir by natural pressures. In some parts of the area, however, there was not enough difference between the pressures of the respective local wells to give the desired range of data with natural pressures. Therefore, it was necessary to obtain the gas injected into two of the intake wells from the discharge of compressor plants in the field gas-gathering systems.

Pressure Gages and Metering Equipment

Pressures were measured with dead-weight pressure gages on five of the wells and with spring gages on one well. Supplemental continuous records of pressure on the wells under different operating conditions were obtained with recording pressure gages.

The rates of gas delivery from the test wells were measured with a critical-flow prover, and the rates of delivery into the wells were measured with orifice meters. A gravity balance was used for measuring the specific gravity of the gas delivered from and into the wells.

Description of Tests and Data

The procedure for conducting the tests on each of the six wells depended upon the operating conditions of the well and data revealed as the study of its characteristics progressed. In general, the data obtained during these tests included a series of working pressures gaged at the wellhead, corresponding rates of gas delivery from or into the well depending upon whether outflow or inflow tests were being conducted, shut-in pressure of the well gaged at the wellhead, specific gravity of the gas, depth of the well, and size and length of flow string in the well. The data for plotting - pressure factors $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$ and the corresponding rates of gas delivery from or into the well - were determined from the observed data. The observations and plotting data obtained from the outflow and inflow tests on the six gas wells studied are given in table 1; these data are supplemented with the following discussion of the testing procedure on each of the wells.

Well A

Well A near Borger, Carson County, Tex., was completed in 1929 with an initial open-flow volume of 22,500,000 cubic feet of gas per 24 hours and a shut-in pressure of 369 pounds per square inch gage.

The shut-in pressure of the well at the wellhead was 298 pounds per square inch gage on August 22 after being shut in for approximately 48 hours. After the shut-in pressure was gaged a back-pressure test was made on the well to determine the pressure-flow relationship for delivery of gas from the well. Working pressures and corresponding rates of gas delivery were gaged under four operating conditions.

The well was closed in after the outflow test, and the following day (August 23) the shut-in pressure on the well was the same (298 pounds per square inch gage) as that observed before the outflow test.

An inflow test was made on the well August 24. Just before this test was begun a shut-in pressure of 298 pounds per square inch gage again was observed. Gas was delivered into the well at four different rates of flow during the inflow test. The gas came from a well about 3 miles from well A and was delivered through part of the gathering system under natural pressure.

The well again was closed in after the injection test, and a shut-in pressure of 299 pounds per square inch gage was observed the following day (August 25).

Shut-in pressures gaged on the well and the behavior of pressure and flow conditions following changes in the operation of the well showed that stabilized conditions were reached in a rather short time and that the data obtained after stabilization periods of about 1-1/2 hours represented conditions of approximate equilibrium of pressures and flow throughout the well system.

TABLE 1. - Pressure and flow data for outflow and inflow tests on gas wells in the Texas Panhandle field

Well	Kind of test	1/ Date of test	Reading no.	Wellhead pressures, pounds per square inch gage		Pressures at sand face, pounds per square inch absolute		2/ Plotting data, $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$, thousands	3/ Q	Specific gravity of gas (air=1)	Time allowed for stabilization of pressure and flow conditions, hours
				Shut-in	Working	Shut-in P_f	Working P_s				
A	Outflow	8/22	1	4/ 298	287	330	319	7.1	872	5/ 0.698	6/ 1-1/2
			2		278		309	13.4	1,518		6/ 1-1/2
			3		267		298	20.1	2,273		6/ 1-1/2
			4		253		283	28.8	3,090		6/ 1-1/2
A	Inflow	8/24	1	7/ 298	311	330	343	8.8	1,033	8/ .681	6/ 1-1/2
			2		324		357	18.6	2,020		6/ 1-1/2
			3		335		369	27.3	2,818		6/ 1-1/2
			4		352		387	40.9	4,060		6/ 1-1/2
B	Outflow	8/28	1	9/ 311	306	341	336	3.4	3,720	8/ .691	6/ 1-1/2
			2		300		330	7.4	6,480		6/ 1-1/2
			3		292		322	12.5	9,850		6/ 1-1/2
			4		280		312	19.0	13,420		6/ 1-1/2
			5		265		298	27.5	17,300		6/ 1-1/2
B	Inflow	8/29	1	7/ 311	318	341	349	5.5	4,900	8/ .688	6/ 1-1/2
			2		322		353	8.3	6,430		6/ 1-1/2
			3		325		355	9.7	7,870		6/ 1-1/2
			4		332		363	15.5	10,700		6/ 1-1/2
			5		347		376	25.1	15,400		6/ 1-1/2
B	Inflow	9/6 to 9/12	1	7/ 310	322	340	353	9.0	7,040	10/ .688	11/ 2/3
			2		321		352	8.3	6,600		11/ 23
			3		332		362	15.4	11,400		11/ 48
			4		329		360	14.0	10,500		11/ 72
			5		324		354	9.7	8,770		11/ 98
			6		324		354	9.7	8,720		11/ 120
			7		327		358	12.6	10,020		11/ 144

TABLE 1. - Pressure and flow data for outflow and inflow tests on gas wells in the Texas Panhandle field - continued.

Well	Kind of test	1/ Date of test	Reading no.	Wellhead pressures, pounds per square inch gage		Pressures at sand face, pounds per square inch absolute		2/ Plotting data, $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$, thousands	3/ Q	Specific gravity of gas (air=1)	Time allowed for stabilization of pressure and flow conditions, hours
				Shut-in	Working	Shut-in P_f	Working P_s				
C	Outflow	9/4	1	12/260	238	290	267	12.8	2,060	10/0.698	13/ 1-1/2
			2		227		255	19.1	2,810		13/ 1-1/2
			3		214		242	25.5	3,620		13/ 1-1/2
			4		200		226	33.0	4,420		13/ 1-1/2
C	Inflow	9/4 to 9/12	3		314	14/290	348	37.0	2,417	8/ .684	11/ 144
			4		318		352	39.8	2,235		11/ 192
C	Inflow	9/23	1	15/268	279	299	310	6.7	1,170	8/ .684	13/ 1-1/2
			2		284		316	10.5	1,630		13/ 1-1/2
			3		295		328	18.2	2,590		13/ 1-1/2
			4		305		339	25.5	3,420		13/ 1-1/2
C	Outflow	9/29 to 10/9	1	16/260	213	290	241	26.0	1,890	8/ .689	11/ 144
			2		207		234	29.3	1,780		11/ 192
D	Outflow	9/18	1	17/295	278	321	304	10.6	4,800	8/ .645	6/ 1-1/2
			2		271		296	15.4	6,110		6/ 1-1/2
			3		263		289	19.5	7,480		6/ 1-1/2
			4		253		279	25.2	8,930		6/ 1-1/2
D	Outflow	9/19 to 9/20	1	17/295	283	321	309	7.5	3,581	8/ .645	1-1/2
			2		277		303	11.2	4,780		1-1/2
			3		275		301	12.4	4,750		18/24-3/4
D	Inflow	9/22	1	17/295	306	321	333	7.9	3,520	8/ .622	6/ 1-1/2
			2		313		340	12.6	5,070		6/ 1-1/2
			3		320		347	17.4	6,500		6/ 1-1/2
			4		334		361	27.3	9,160		6/ 1-1/2
			5		346		373	36.1	11,730		6/ 1-1/2

TABLE 1. - Pressure and flow data for outflow and inflow tests on gas wells in the Texas Panhandle field - continued

Well	Kind of test	1/ Date of test	Reading no.	Wellhead pressures, pounds per square inch gage		Pressures at sand face, pounds per square inch absolute		2/Plotting data, $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$, thousands	3/Q	Specific gravity of gas (air=1)	Time allowed for stabilization of pressure and flow conditions, hours
				Shut-in	Working	Shut-in P_f	Working P_s				
D	Inflow	9/23	1	1295	312	321	339	11.9	4,950	8/ 0.622	1-1/2
		to	2		315		342	14.0	4,870	.622	19/23-1/2
		9/26	3		322		349	18.8	6,650	.622	1-1/2
			4		338		365	30.2	10,150	.622	1-1/2
			5		339		366	30.9	9,570	.640	19/23-1/2
			6		339		366	30.9	9,580	.640	19/48
E	Outflow	8/3	1	1292	253	318	279	23.3	9,900	10/ .640	6/ 1-1/2
			2		260		286	19.3	8,500		6/ 1-1/2
			3		267		292	15.8	7,700		6/ 1-1/2
			4		279		305	8.1	3,700		6/ 1-1/2
			5		275		300	11.1	5,250		6/ 1-1/2
E	Inflow	8/9	1		360	14/318	388	49.4	10,630	10/ .640	6/ 1-1/2
			2		349		376	40.3	8,970		6/ 1-1/2
			3		336		364	31.4	7,170		6/ 1-1/2
			4		321		348	20.0	4,660		6/ 1-1/2
E	Outflow	8/11	3		235	14/318	260	33.5	9,960	10/ .640	6/ 1-1/2
			4		222		246	40.6	11,200		6/ 1-1/2
			5		244		269	28.7	8,570		6/ 1-1/2
			6		264		289	17.5	5,940		6/ 1-1/2
E	Outflow	208/12	1		248	14/318	274	26.0	12,520	10/ .640	6/ 1-1/2
			2		254		280	22.7	10,730		6/ 1-1/2
			3		260		286	19.3	9,130		6/ 1-1/2
			4		226		253	37.1	15,630		6/ 1-1/2
			5		272		297	12.9	6,120		6/ 1-1/2

TABLE 1. - Pressure and flow data for outflow and inflow tests on gas wells in the Texas Panhandle field - continued

Well	Kind of test	1/Date of test	Reading no.	Wellhead pressures, pounds per square inch gage		Pressures at sand face, pounds per square inch absolute		2/Plotting data, $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$, thousands	3/Q	Specific gravity of gas (air=1)	Time allowed for stabilization of pressure and flow conditions, hours
				Shut-in	Working	Shut-in P_f	Working P_s				
E	Inflow	8/13	1		348	14/318	374	38.8	14,180	10/0.640	6/ 1-1/2
			2		339		366	32.9	12,200		6/ 1-1/2
			3		329		356	25.6	9,800		6/ 1-1/2
			4		317		344	17.2	6,870		6/ 1-1/2
F	Outflow	8/15	1	21/229	167	257	194	28.4	14,600	10/ .700	6/ 1-1/2
			2		200		227	14.35	9,600		6/ 1-1/2
			3		206		234	11.2	8,400		6/ 1-1/2
			4		216		244	6.5	5,900		6/ 1-1/2
			5		184		212	21.1	12,200		6/ 1-1/2
F	Inflow	8/16 and 8/17	1		255	14/257	284	14.7	8,500	8/ .665	6/ 1-1/2
			2		283		313	32.0	14,400		6/ 1-1/2
			3		331		361	64.3	23,000		6/ 1-1/2
			4		378		410	102.1	31,300		6/ 1-1/2
			5		438		471	155.8	42,000		6/ 1-1/2
			6		280		309	29.5	14,200		6/ 1-1/2

TABLE 1. - Pressure and flow data for outflow and inflow tests on gas wells in the Texas Panhandle field - continued

Miscellaneous data	Well					
	A	B	C	D	E	F
Depth of first sand --- feet	2,425	2,095	2,460	1,885	1,828	2,350
Depth of last sand --- do.	2,630	2,300	2,690	---	1,947	2,715
Total depth --- do.		2,300	2,765	1,905	1,947	3,077
Size of casing --- inches	8-1/4	8-1/4	8-1/4	8-1/4	8-1/4	10
Depth casing set --- feet	2,163	2,034	22/2,817	1,775	1,828	1,893
Size of liner --- inches	6-5/8	---	---	---	---	---
Depth liner set --- feet	2,410	---	---	---	---	---

1/ All data obtained in 1934. 2/ $P_f^2 - P_s^2$ for outflow test and $P_s^2 - P_f^2$ for inflow test. 3/ Q in thousands of cubic feet of gas per 24 hours at pressure of 14.4 pounds per square inch and temperature of 60°F. 4/ Well shut in for approximately 48 hours. 5/ Specific gravity determined several days before test. 6/ Pressure and flow conditions stabilized. 7/ Observed just before test. 8/ Specific gravity determined during test. 9/ Well shut in for 7 days. 10/ Assumed specific gravity. 11/ These stabilization periods represent cumulative lapsed time. 12/ Well shut in for approximately 3 days. 13/ Pressure and flow conditions not stabilized. 14/ Based upon previous outflow test. 15/ Well shut in for 16 days. 16/ Based upon studies of shut-in pressure record. 17/ Well shut in for approximately 1 month. 18/ Operating condition for reading no. 2 allowed to continue to 24-3/4 hours for reading no. 3. 19/ Operating condition for reading no. 1 allowed to continue to 23-1/2 hours for reading no. 2; operating condition for no. 4 allowed to continue to 23-1/2 hours for reading no. 5 and then to 48 hours for reading no. 6. 20/ Well open-flowed and cavings removed before this test. 21/ Well closed in for approximately 2 weeks. 22/ Casing ripped 2,680 to 2,690 feet.

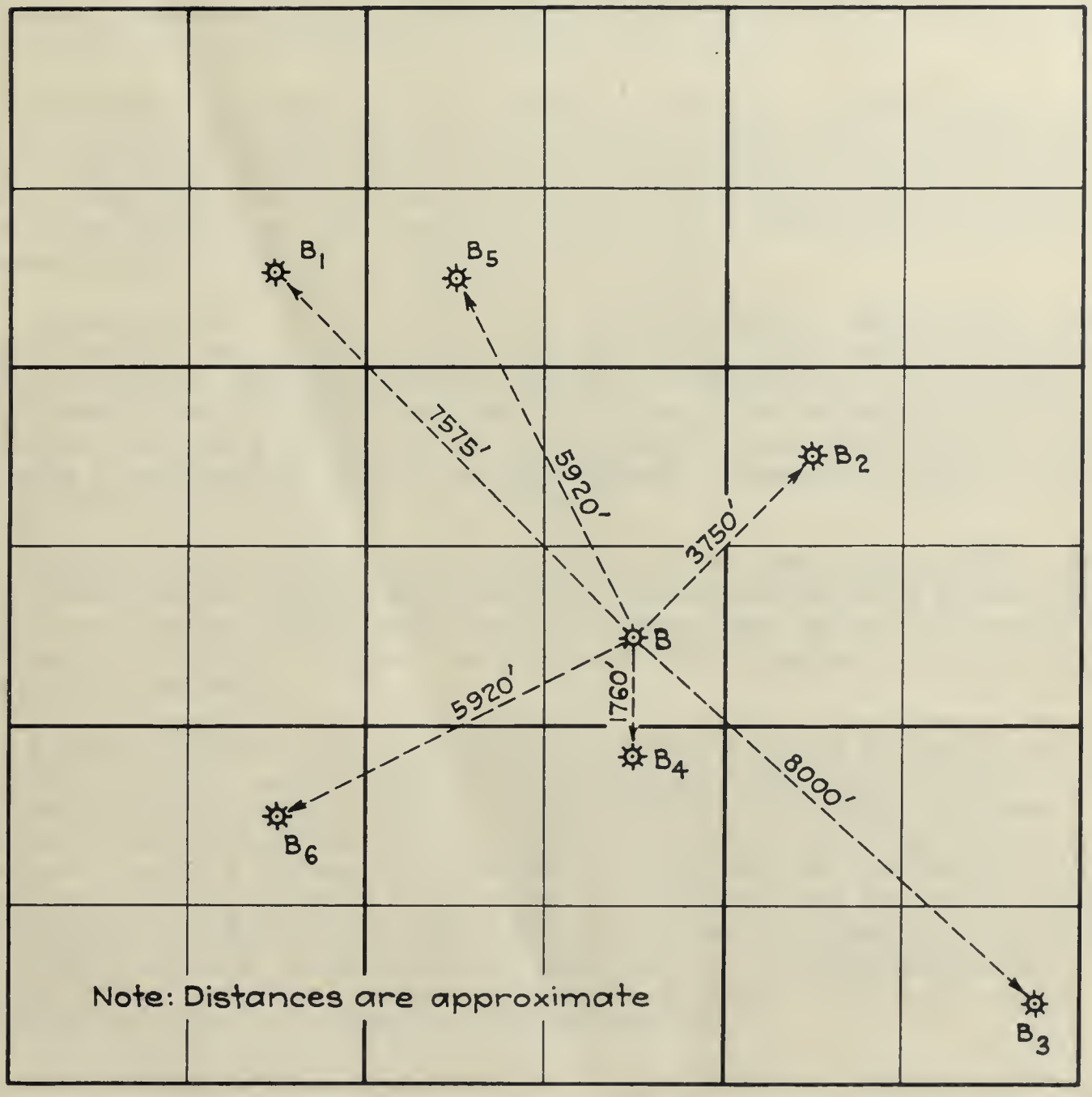


Figure 1.— Relative locations of well B and nearby wells.

Well B

Well B near Skellytown, Carson County, Tex., was completed in 1930 with an initial open-flow volume of 53,000,000 cubic feet of gas per 24 hours and a shut-in pressure of 375 pounds per square inch gage.

The shut-in pressure of well B gaged just before an outflow test on August 28, after the well had been closed in for 7 days, was 311 pounds per square inch gage. During the outflow test wellhead working pressures and delivery rates were observed under five different operating conditions. The shut-in pressure on the well, gaged 40 minutes after completion of the back-pressure test, was 312 pounds per square inch gage.

An inflow test was made on the well the following day (August 29). The shut-in pressure on the well, gaged just before this test, was 311 pounds per square inch gage. Wellhead working pressures corresponding to five rates of gas delivery were observed during the inflow test. The relation of the shut-in pressure of well B to that of others connected to the same gathering system made it necessary to deliver the gas through a compressor plant to obtain the desired range of pressure and flow data.

Pressure and flow data recorded during both the outflow and inflow tests were observed after stabilization periods of approximately 1-1/2 hours. The behavior of pressures and flow in this well showed that approximate equilibrium was reached in the well system 2 or 3 minutes after changes in the operating condition of the well.

A more detailed study of the intake capacity of well B was made from September 6 to 12. Approximately 57,000,000 cubic feet of gas was delivered into the well during this period at varying rates of flow, depending upon changes in the pressure maintained on the discharge line from the compressor station - between a minimum of approximately 6,500,000 and a maximum of approximately 11,500,000 cubic feet of gas per 24 hours. The shut-in pressure gaged on the well September 6 before the inflow test was begun was 310 pounds per square inch gage; on September 12, approximately 24 hours after completion of the inflow test, it was 311 pounds per square inch gage; and on September 15 it was 310 pounds per square inch gage.

The location of the wells in the area immediately surrounding well B is shown in figure 1. Pressures were gaged daily from September 6 to 12 on wells B_1, B_2, and B_3, which were shut in before the inflow test on well B was begun. Blank plates were inserted in the orifice-meter flanges before the test to insure that the data would be representative of changes in formation pressures and would not be influenced by leakage through wellhead connections. These data are shown in table 2. Gas was produced from wells B_4 and B_5 during the period of the inflow test at rates of approximately 20,000,000 and 12,000,000 cubic feet of gas per 24 hours, respectively.

Well C

Well C in Gray County several miles south of Pampa, Tex., was plugged back to the gas horizon in 1928. Its initial open flow after the 8-1/4-inch casing was ripped at the gas horizon was reported as 18,000,000 cubic feet of gas per 24 hours.

An outflow test was made on well C September 4. Working pressures and delivery data based on stabilization periods of approximately 1-1/2 hours were obtained under four operating conditions. The well had been closed in September 1, and the shut-in pressure gaged September 4 just before the flow test was begun was 260 pounds per square inch gage.

TABLE 2. - Behavior of shut-in pressures on gaswells surrounding well B during an inflow test on well B

Date[1/]	Shut-in pressure, pounds per square inch gage		
	Well B_1	Well B_2	Well B_3
9/6	298-3/4	310-1/2	319-1/4
9/7	299-1/2	310-1/2	319-1/4
9/8	299-3/4	310-1/2	319-1/2
9/9	299-3/4	310-1/2	319-1/2
9/10	300	310-1/2	319-1/2
9/11	300-1/2	310-3/4	319-1/2
9/12	300-1/2	310-3/4	319-1/2
9/13	300-1/2	310-3/4	319-1/2

1/ Data obtained in 1934.

Note. - Well B_1 was shut in September 6, 3-1/2 hours before the shut-in pressure gaged on that date was observed.
Wells B_2 and B_3 were shut in August 27, 1934.
Delivery of gas into well B was discontinued September 12, and the well remained closed in September 13.

The stabilized shut-in pressure of well C was not determined after the outflow test, as an inflow test was begun approximately 25 minutes after the outflow test was completed. During this 25-minute interval the pressure on the well increased from 200 to approximately 247 pounds per square inch gage.

The working pressure and rate of delivery were observed after a stabilization period of approximately 1-1/2 hours. Delivery of gas into the well was continued throughout the period September 4 to 12. The gas was delivered into well C under natural pressure from a well several miles west completed in the same horizon as the "pay" formation of well C. The well from which the gas was obtained was characterized by slow stabilization of pressure and flow conditions; as a result, the rate at which the supply of gas was available at constant pressure diminished slowly throughout the test.

The behavior of pressures and flow in well C represented a very slow type of stabilization. Even though the rate of flow of gas into the well was decreasing, the working pressure continued to increase during the period of

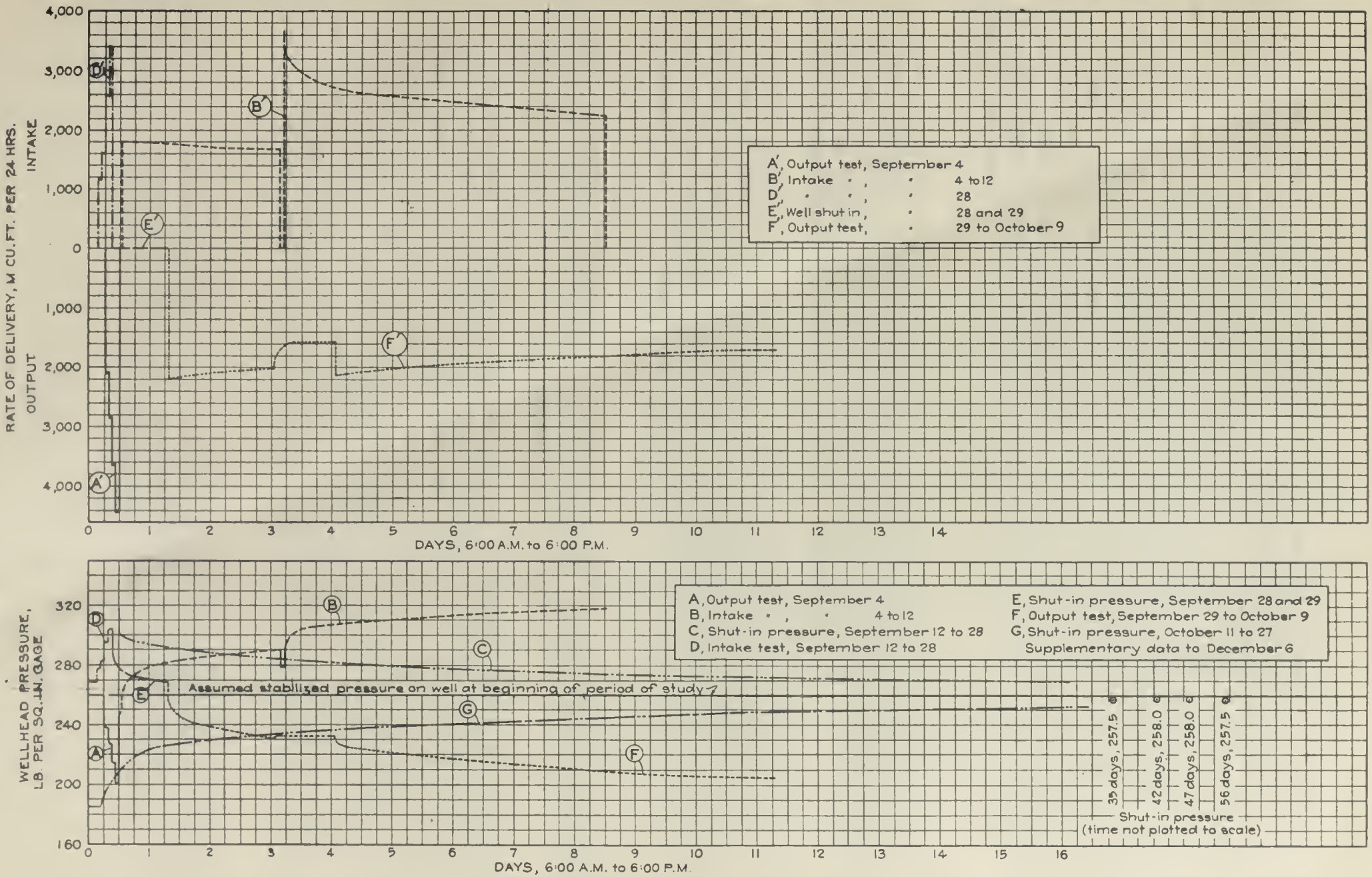

RATE OF DELIVERY, M CU. FT. PER 24 HRS.
INTAKE
OUTPUT
4,000
3,000
2,000
1,000
0
1,000
2,000
3,000
4,000
A', Output test, September 4
B', Intake " , " 4 to 12
D', " " , " 28
E', Well shut in, " 28 and 29
F', Output test, " 29 to October 9
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14
DAYS, 6:00 A.M. to 6:00 P.M.
WELLHEAD PRESSURE, LB PER SQ. IN. GAGE
320
280
240
200
160
A, Output test, September 4
B, Intake " , " 4 to 12
C, Shut-in pressure, September 12 to 28
D, Intake test, September 12 to 28
E, Shut-in pressure, September 28 and 29
F, Output test, September 29 to October 9
G, Shut-in pressure, October 11 to 27
Supplementary data to December 6
Assumed stabilized pressure on well at beginning of period of study
35 days, 257.5 @
42 days, 258.0 @
47 days, 258.0 @
56 days, 257.5 @
Shut-in pressure
(time not plotted to scale)
0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
DAYS, 6:00 A.M. to 6:00 P.M.

Figure 2.—Rates of gas deliveries and pressures observed during tests on well C.

the test. The initial rate of inflow of gas was approximately 1,800,000 cubic feet per 24 hours (September 4). The rate at which gas was available diminished gradually during the ensuing period; on September 7 it was approximately 1,690,000 cubic feet per 24 hours. During this period the working pressure on well C increased from 270 to 291 pounds per square inch gage. The rate of delivery of gas into the well was increased September 7 to 3,100,000 cubic feet per 24 hours. Under this operating condition the supply of gas available diminished rather rapidly, and 5 days later (September 12) the rate of flow into well C had decreased to 2,235,000 cubic feet per 24 hours. Pressures and corresponding measurement data after stabilization periods of approximately 6 and 8 days, respectively, are shown in table 1. Variations in the working pressure and in the rate of delivery of gas into the well during the period September 4 to 12 are shown, respectively, by lines B and B', figure 2.

The well was closed in at the end of the inflow test September 12, and the shut-in pressure on the well was gaged daily from September 12 to 28. The decrease of the shut-in pressure on the well from 318 pounds per square inch gage at the termination of the inflow test (Sept. 12) to 268 pounds per square inch September 28 is shown by line C, figure 2.

A second inflow test was made on well C September 28. During this test the data were observed after stabilization periods of approximately 1-1/2 hours and included working pressures and corresponding measurement data obtained under four operating conditions. The calculations were based upon a shut-in pressure of 268 pounds per square inch obtained before the inflow test.

Well C was shut in September 28 after the inflow test, and the working pressure decreased from 305 pounds per square inch gage at the close of the inflow test to 269 pounds on the following day (Sept. 29).

An outflow test lasting 12 days was begun September 29 to obtain outflow data based upon pressure and flow stabilization similar to the stabilization that existed during the inflow test made September 4 to 12. Working pressures and gas-delivery data were observed at intervals from September 29 to October 9. Variation of the working pressure and the rate of delivery during this period are shown, respectively, by F and F', figure 2. The test was continued until October 11, when the well was closed in. The rate of gas delivery from the well varied considerably from October 9 to 11 because of external influences, and the data obtained during this interval were of no value.

After the well was closed in on October 11 shut-in pressures gaged from October 12 to December 6 showed that the stabilized pressure on well C was 258 pounds per square inch gage from November 22 to 27. The behavior of the shut-in pressure on the well from October 11 to December 6 is shown by G, figure 2.

Well D

The initial open flow of well D in the Shamrock field, Wheeler County, Tex., was reported to be 47,000,000 cubic feet of gas per 24 hours and the initial shut-in pressure, 400 pounds per square inch gage.

The shut-in pressure observed on well D September 16, after it had been closed in for approximately 1 month, was 295 pounds per square inch gage. It was reported that entrained material from the formation was produced with gas from several of the wells in the vicinity of well D. Therefore well D was open-flowed September 17 to insure that later flow tests would not be affected by cavings or cuttings from the formation in the well bore.

An outflow test was made on well D September 18. Well pressures and corresponding rates of delivery based upon stabilization periods of approximately 1-1/2 hours were obtained under four operating conditions.

A second outflow test was begun on well D September 19. Well pressures and corresponding delivery rates based upon stabilization periods of approximately 1-1/2 hours were obtained under two operating conditions. The delivery of gas under the second operating condition was continued for approximately 24 hours.

Some difficulty was experienced in connecting the critical-flow prover to the "blow-off" fitting above the top gate valve on the well. The prover, therefore, was not disconnected from the wellhead fittings while the well was closed in between the outflow tests made September 18 and 19. Considerable gas leaked through the valves and escaped from the prover during the interval between the two outflow tests when valves a, b, and c (fig. 3) were closed. Nevertheless, the shut-in pressure gaged on well D before the outflow test September 19 was 295 pounds per square inch gage, or the same as that observed when blow-off fitting d was plugged. A blind plate was inserted in the pipe-line connection from the well to eliminate the effect of possible leakage from the pipe line (line pressure, 360 pounds per square inch gage) during the outflow test on September 19 and on later measurements of shut-in pressures. The prover was removed and the blow-off vent was plugged on September 20, after completion of the second outflow test. The shut-in pressure on the well the following day (Sept. 21) was 295 pounds per square inch gage.

An inflow test was made on the well September 22. Before this test was begun the line connecting the well to the gathering system was disconnected at the well, and the gathering system was opened to the atmosphere at this point. The discharged gas stream indicated that the lines of the gathering system near well D were relatively free from foreign materials that might be carried into the well bore during an inflow test. Well pressures and corresponding rates of delivery of gas into the well, based upon stabilization periods of approximately 1-1/2 hours, were obtained under five operating conditions.

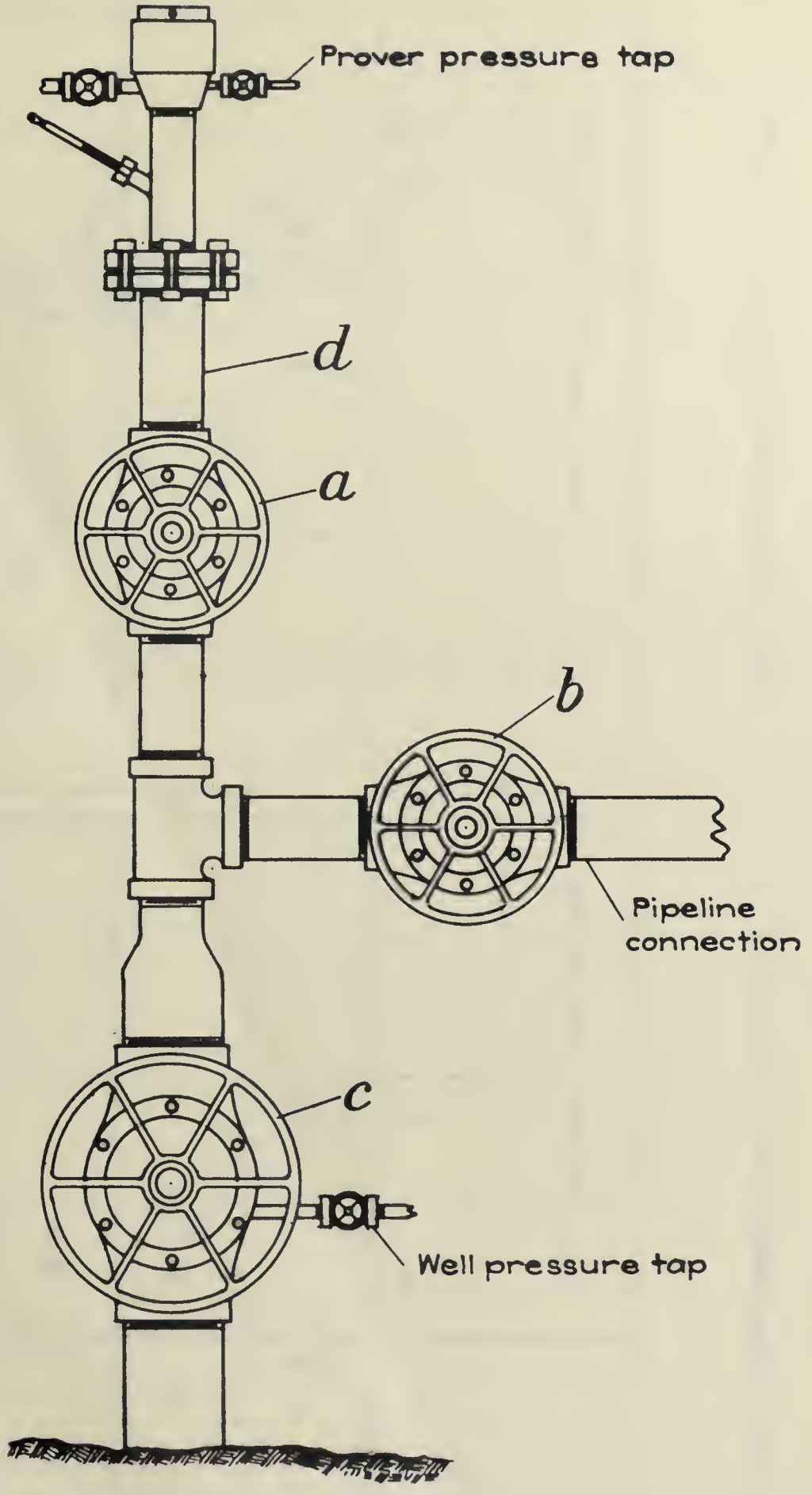


Figure 3.— Wellhead fittings, well D.

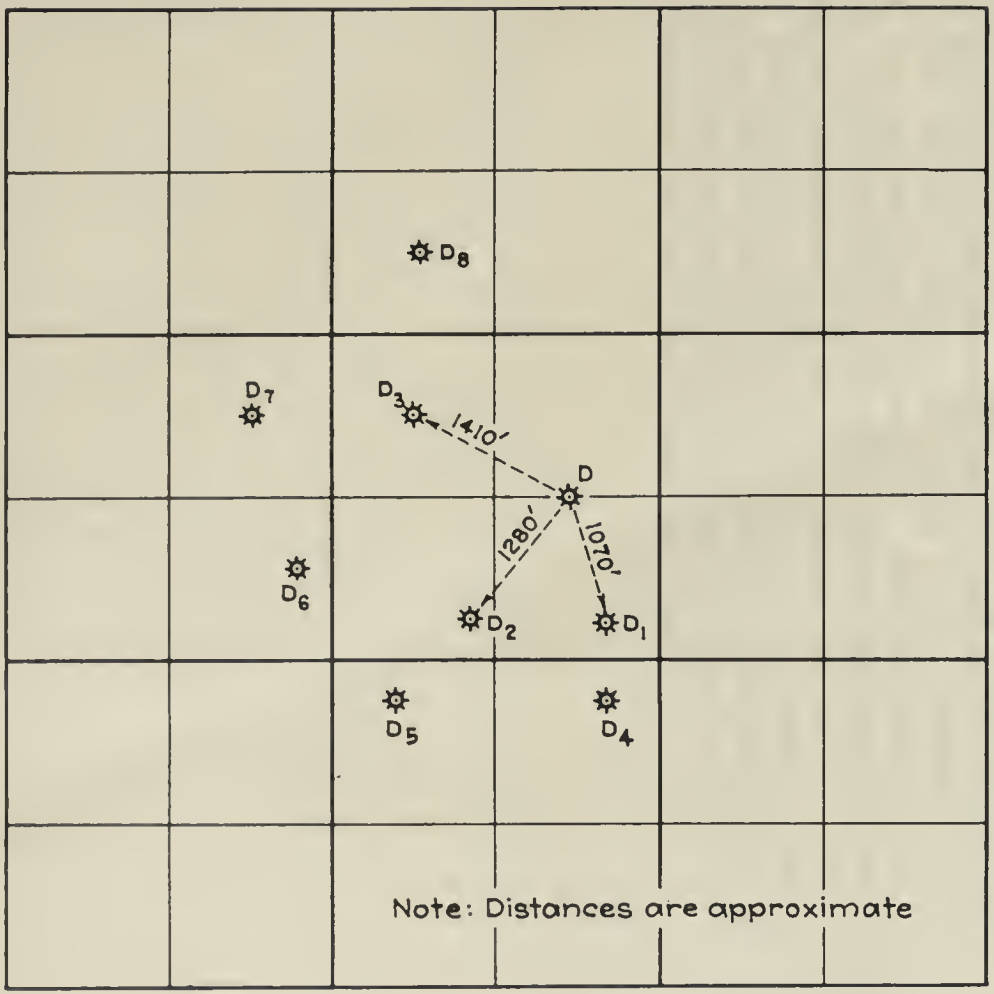


Figure 4.— Relative locations of well D and nearby wells.

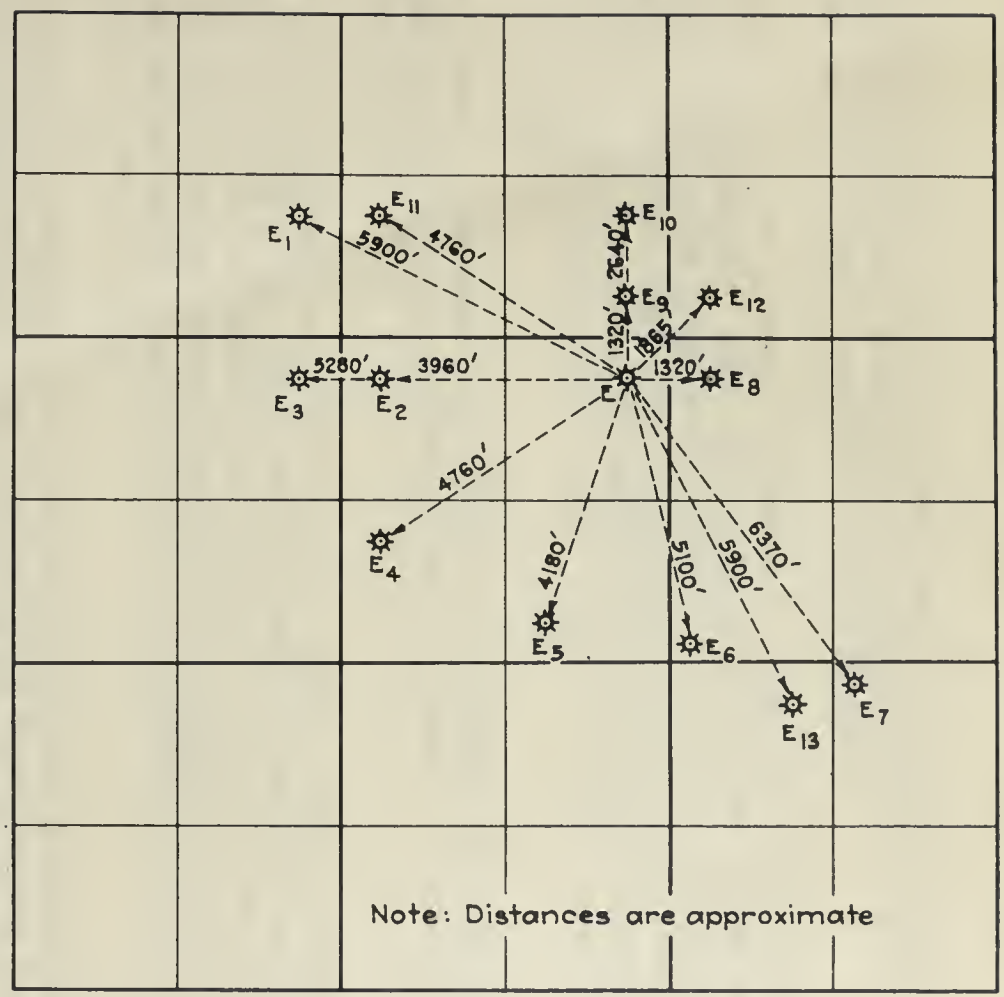
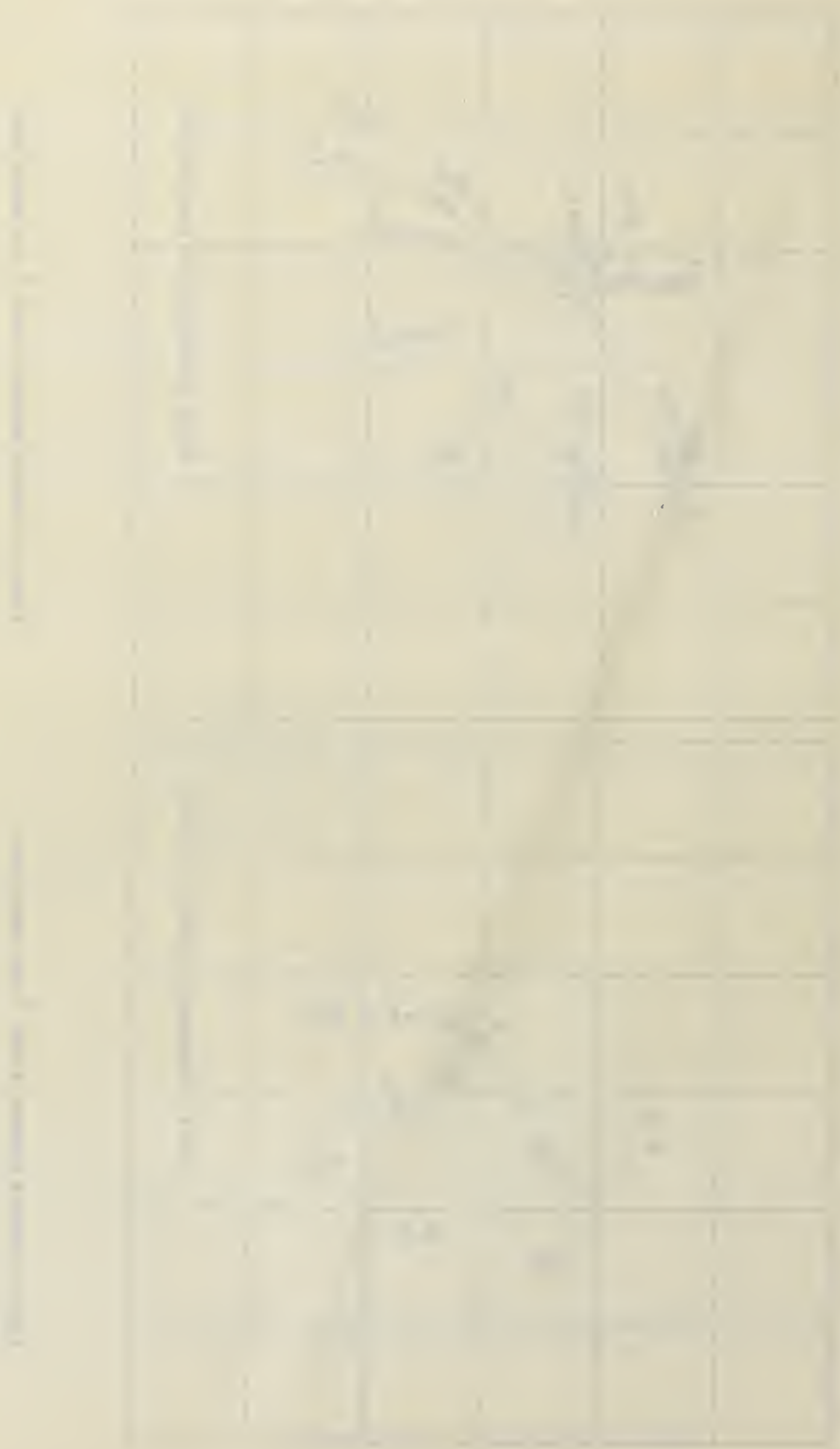


Figure 5.— Relative locations of well E and nearby wells.



The shut-in pressures on the well gaged before the inflow test September 22 and the following day, before the second inflow test, were 294-3/4 and 295-1/4 pounds per square inch gage, respectively.

An inflow test was begun September 23, and well pressures and corresponding rates of gas delivery into the well were obtained under three operating conditions. Data based upon stabilization periods of 1-1/2 hours and 23-1/2 hours during the first operating conditions, upon a stabilization period of 1-1/2 hours during the second operating condition, and upon stabilization periods of 1-1/2, 23-1/2, and 48 hours under the third operating condition are shown in table 1.

The second inflow test was completed September 26. The shut-in pressure gaged on the well the following day (September 27) was 297 pounds per square inch gage.

The relative locations of well D and nearby wells is shown in figure 4. Wells D_1, D_2, D_3, D_4, D_5, and D_8 were closed in, and gas was being produced from wells D_6 and D_7 throughout the period of the outflow and inflow tests of well D. The shut-in pressures of wells D_1, D_2, and D_3 were gaged daily during the period of the tests of well D (September 19 to 27). These pressure data are shown in table 3.

The pipe-line connections to wells D_1, D_2, and D_3 were not blanked off with blind plates. However, it is improbable that leakage through the closed valves on the pipe-line connections to the wells materially affected the relation of the shut-in pressures gaged on the respective wells during the period of the inflow test of well D. Although the pressure on the pipe-line connection to these wells, normally about 200 pounds per square inch gage, was maintained between 300 and 380 pounds per square inch gage during the period of the pressure study, the change occurred 2 days before the initial shut-in pressures on wells D_1, D_2, and D_3 were observed; it is believed that later changes in the shut-in pressures of these wells were representative of pressure changes in the local reservoir zone.

TABLE 3. - Behavior of shut-in pressure of gas wells surrounding well D during outflow and inflow tests of well D.

Date[1]	Pressure on Well D pounds per square inch gage	Shut-in pressure, pounds per square inch gage		
		Well D_1	Well D_2	Well D_3
9/19[2]	276	294-1/2	296-1/4	295-1/4
9/20[3]	275	293-3/4	296	295
9/21[4]	294-1/2	295	296	294-3/4
9/22[5]	294-3/4	295	296-1/4	295-1/4
9/24[6]	315	297-3/4	297-1/2	295-3/4
9/25[7]	339-1/2	300	299	296-1/4
9/26[8] (11 a.m.)	339	300-1/2	298-3/4	296-3/4
9/26 (2:30 p.m.)	293	299-3/4	298-1/2	297
9/27[9]	296-3/4	297	298-1/4	296-1/2

1/ Data were obtained in 1934.

2/ Pressures were observed approximately 5 hours after the outflow test was begun. The rate of delivery from well D was approximately 4,760,000 cubic feet of gas per 24 hours.

3/ Pressures were observed 23-1/2 hours after the outflow test was begun. The rate of delivery from well D was approximately 4,750,000 cubic feet of gas per 24 hours.

4/ Outflow test was terminated the preceding day, and well D had been closed in 21 hours.

5/ Well D had been closed in 48 hours.

6/ These pressures were observed approximately 24 hours after inflow test was begun. The rate of gas delivery into well D was approximately 4,870,000 cubic feet of gas per 24 hours.

7/ Pressures were observed after gas had been delivered into well D for 24 hours at the rate of approximately 9,500,000 cubic feet per 24 hours.

8/ Pressures were observed (11 a.m.) after gas had been delivered into well D for 48 hours at the rate of approximately 9,500,000 cubic feet per 24 hours. The delivery of gas into well D was terminated at 12 noon and the pressures on the wells were gaged approximately 2-1/2 hours later (2:30 p.m.).

9/ Pressures were observed approximately 24 hours after the inflow test was completed.

Well E

On August 3 the shut-in pressure of well E near well D in the Shamrock field, Wheeler County, Tex., was 292 pounds per square inch gage. A back-pressure or outflow test was made on the well to determine its producing characteristic. Wellhead pressures and corresponding rates of gas delivery after stabilization periods of approximately 1-1/2 hours, were observed under five operating conditions.

Gas was delivered into well E at a rate of approximately 11,000,000 cubic feet per 24 hours from August 5 to 9. The rate of gas delivery into the well was changed August 9, and well pressures and corresponding rates of gas delivery after relatively brief periods of stabilization were observed under four operating conditions. The inflow test was terminated August 10, and the well was closed in. A shut-in pressure of 295 pounds per square inch gage was observed on the well after it had been closed for approximately 15 hours.

A second outflow test was made on well E August 11. Pressure and flow data, based upon relatively short periods of stabilization, were obtained under four operating conditions.

Well E was open-flowed August 11 after completion of the outflow test, and some foreign material similar to cavings from the walls of the open hole or cuttings from the productive horizon was removed from the well. A third outflow test was made on the well August 12 after it had been "blown."

A second inflow test was made on well E August 13. Pressure and flow data based upon relatively short periods of stabilization were obtained under four operating conditions.

The gas was delivered into well E under natural pressure from other wells in the Shamrock field.

Shut-in pressures were gaged daily on 10 closed-in wells in the area around well E from August 4 to 12; these data are shown in table 4. The relative locations of well E and surrounding wells are shown in figure 5. Pressure data were not obtained on wells E_{11}, E_{12}, and E_{13}, as they were being operated and their respective working pressures were influenced more by changes in pressures on the gathering system than by relatively small local changes in the reservoir pressure.

TABLE 4. - Behavior of shut-in pressures on gas wells surrounding well E during outflow and inflow tests on well E

Date 1/	Shut-in pressures, pounds per square inch gage										
	Well E	Well E_1	Well E_2	Well E_3	Well E_4	Well E_5	Well E_6	Well E_7	Well E_8	Well E_9	Well E_{10}
8/3	292										
8/4		293	295	295	296	297	295	294	293	293	293
8/5		294	295	295	298	298	295	294	293	293	293
8/6		294	296	295	298	298	297	296	295	296	294
8/7		294	295	296	297	298	296	296	295	295	293
8/8		294	295	296	298	297	297	296	295	296	293
8/9		293	296	296	298	298	297	297	295	296	293
8/10		293	295	296	298	297	297	297	295	295	293
8/11	295	293	295	295	297	296	297	297	294	294	293
8/12	293	293	295	295	297	296	296	297	293	293	292

1/ Data obtained in 1934.

Note. - An outflow test was made on well E August 3 .
Gas was delivered into well E from August 5 to 9 at an approximate rate of 11,000,000 cubic feet per 24 hours and at lower rates on August 9. The working pressure on the well during this period was about 360 pounds per square inch gage.

An outflow test was made on well E August 11; after completion of this test the well was opened to the atmosphere for 25 minutes.

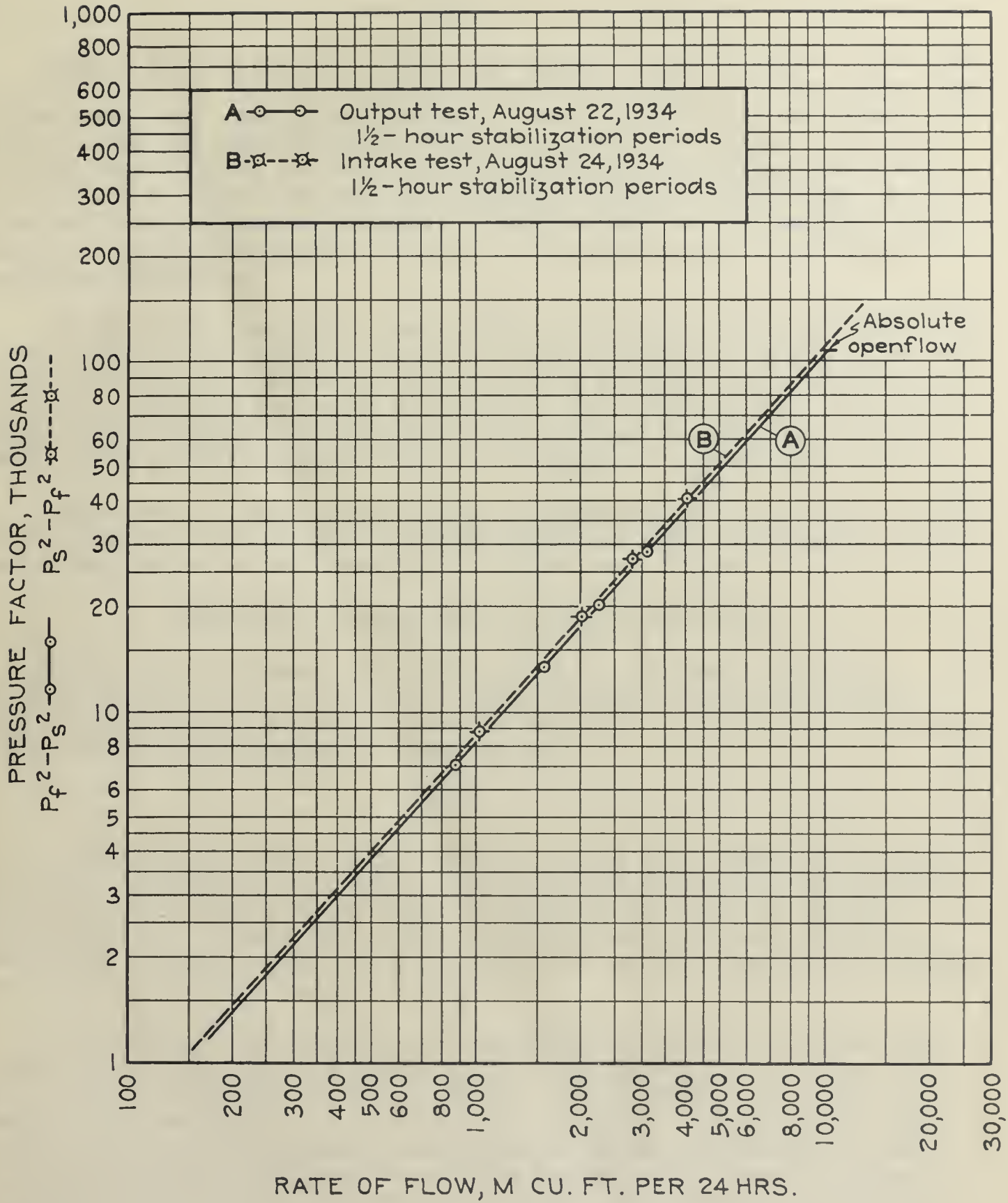


Figure 6.— Comparison of output and intake characteristics of well A.

Well F

Well F was in Hutchinson County near Fritch, Tex. On August 15 the shut-in pressure on the well after being closed in for 2 weeks was 229 pounds per square inch gage. The well was opened to the atmosphere for approximately 10 minutes after the shut-in pressure had been gaged. The stream of gas from the well showed little evidence of entrained liquid or cavings and it was assumed that the well bore was relatively free from such materials. An outflow test then was made on the well. Pressures and rates of gas delivery, based upon relatively short periods of stabilization, were observed under five operating conditions.

An inflow test was made of well F August 16 and 17. Pressures and rates of gas delivery were observed under five operating conditions August 16, and a check test was made August 17 under pressure and flow conditions similar to those included in the inflow test made on the preceding day. The gas returned to the formation through well F was produced from wells in the Panhandle field and was delivered from the discharge of a compressor plant.

The behavior of well pressures and gas-delivery rates during the tests of well F showed that pressure and flow conditions were approximately stabilized within an hour after operating conditions were changed.

Comparison of Output and Intake Characteristics

The output and intake characteristics of the gas wells tested are shown graphically by plotting rates of gas delivery on logarithmic paper against corresponding values of pressure factors $P_f^2 - P_s^2$ (output characteristic) and $P_s^2 - P_f^2$ (intake characteristic).

Well A

The output and intake characteristics of well A are compared in figure 6. Line A represents the producing characteristic of the well obtained by plotting rates of gas delivery against corresponding values of pressure factor $P_f^2 - P_s^2$. Line B represents the intake characteristic of the well obtained by plotting rates of gas delivery into the well against corresponding values of pressure factor $P_s^2 - P_f^2$.

The slopes of lines A and B are approximately the same throughout the range of data. For the same values of pressure factors $P_f^2 - P_s^2$ and $P_s^2 - P_f^2$, the rate of gas delivery from the well is approximately 4 percent greater than the rate of gas delivery into the well. The behavior of the pressures and delivery rates following changes in the operating condition of well A indicated that pressure and flow conditions in the well system were nearly stable after 1-1/2 hours; therefore, the results of the tests represent approximately the stabilized outflow and inflow characteristics of the well.

Well B

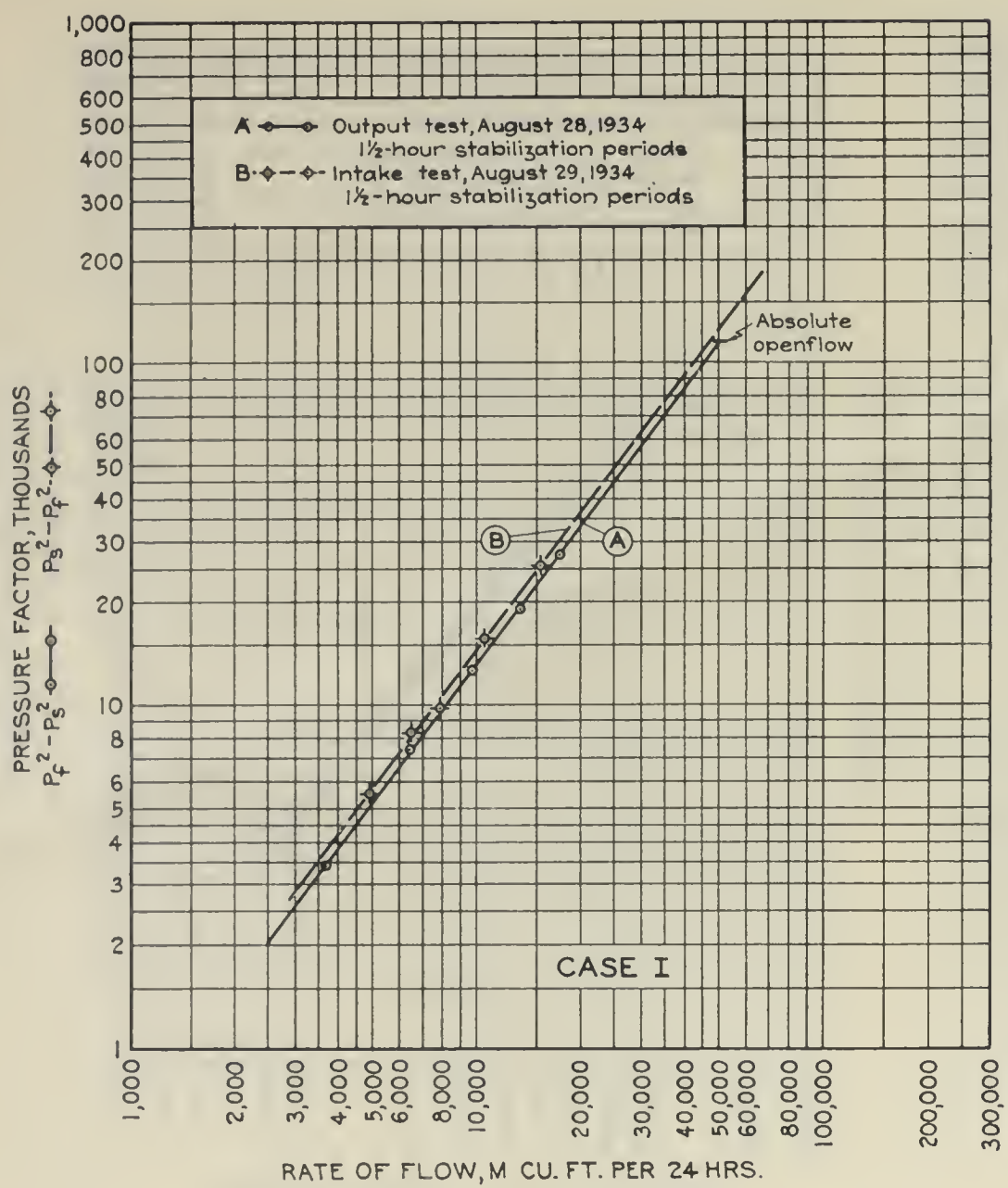
The output and intake characteristics of well B are shown graphically in figure 7. Lines A and B represent the output and intake characteristics of the well. The plotted data under case I are based upon stabilization periods of approximately 1-1/2 hours; under this condition the slopes of the lines representing the output and intake characteristics of the well are virtually the same throughout the range of data, and for equal values of the respective pressure factors the rate of gas delivery from the well is about 7 percent greater than the rate of delivery into the well.

The results of an inflow test based upon long periods of pressure and flow stabilization (test was begun September 6 and completed September 12) are shown under case II, figure 7. The rate of gas delivery into well B during this period ranged from 6,600,000 to 11,400,000 cubic feet per 24 hours. The plotted data based upon observations during the initial 24-hour period corresponded closely with the data based upon the inflow test of August 29. As the delivery of gas into the well was continued, there was a general increase in the rate of gas flow for a definite value of the pressure factor. (Note positions of consecutive points 1 to 7 inclusive, case II, with respect to line B.)

Well C

Well C was characterized by very slow stabilization of pressure and flow conditions. Line A, case I, figure 8, was determined from data observed after stabilization periods of 1-1/2 hours during the inflow test made September 4. The pressure factor is based upon a formation pressure of 290 pounds per square inch absolute calculated from the shut-in pressure of 260 pounds per square inch gage at the wellhead observed before the test. Line B, case I, represents the intake characteristic of the well and is based upon operating pressures observed after stabilization periods of approximately 1-1/2 hours; this test was made September 28. The relationship represented by graph B is based upon pressure-factor values calculated from the shut-in pressure of 268 pounds per square inch gage observed on the wellhead September 28 just before the inflow test was started. Lines A and B, case I, have approximately equal slopes and show that for the same values of pressure factors $P_f^2 - P_s^2$ and $P_s^2 - P_f^2$ the rate of gas delivery from the well is about 6.5 percent greater than the rate of delivery into the well.

Data determined from observations made 6 and 8 days after outflow and inflow tests were begun September 29 and 4, respectively, are plotted under case II, figure 8. As the operating condition of the well was changed by external influence during the stabilization periods of both the outflow and inflow conditions of operation, the degree of stabilization of pressure and flow conditions is not exactly the same for the two tests, even though the lapsed time in each of the tests is the same. Although the outflow and inflow data are not exactly comparable because of the probable discrepancy in stabilization, there is a marked similarity in the reduced rates of gas delivery from the well and gas delivery into the well as the operations were


A Output test, August 28, 1934
1½-hour stabilization periods
B Intake test, August 29, 1934
1½-hour stabilization periods
Absolute openflow
CASE I
PRESSURE FACTOR, THOUSANDS
$P_f^2-P_s^2$ $P_s^2-P_f^2$
RATE OF FLOW, M CU. FT. PER 24 HRS.

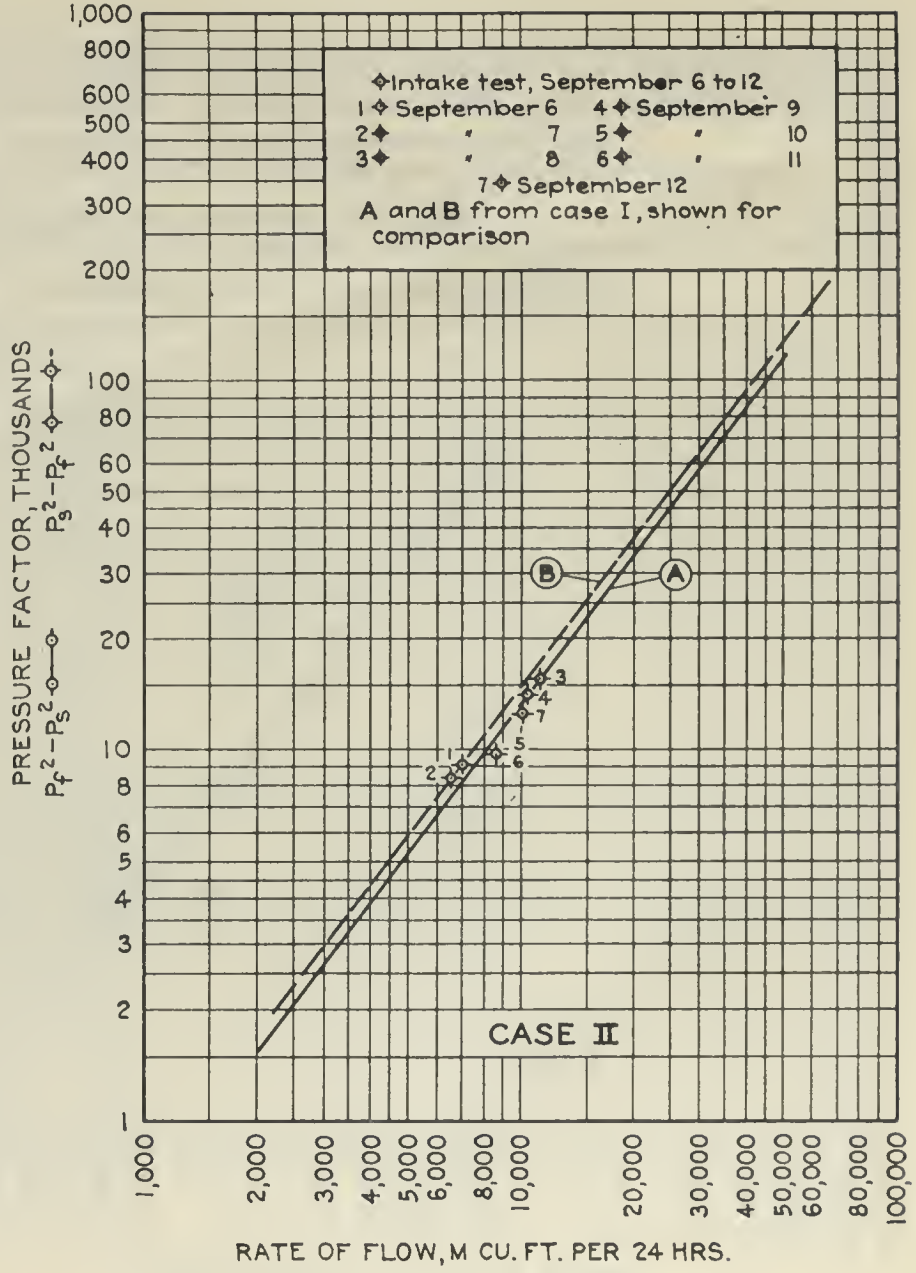
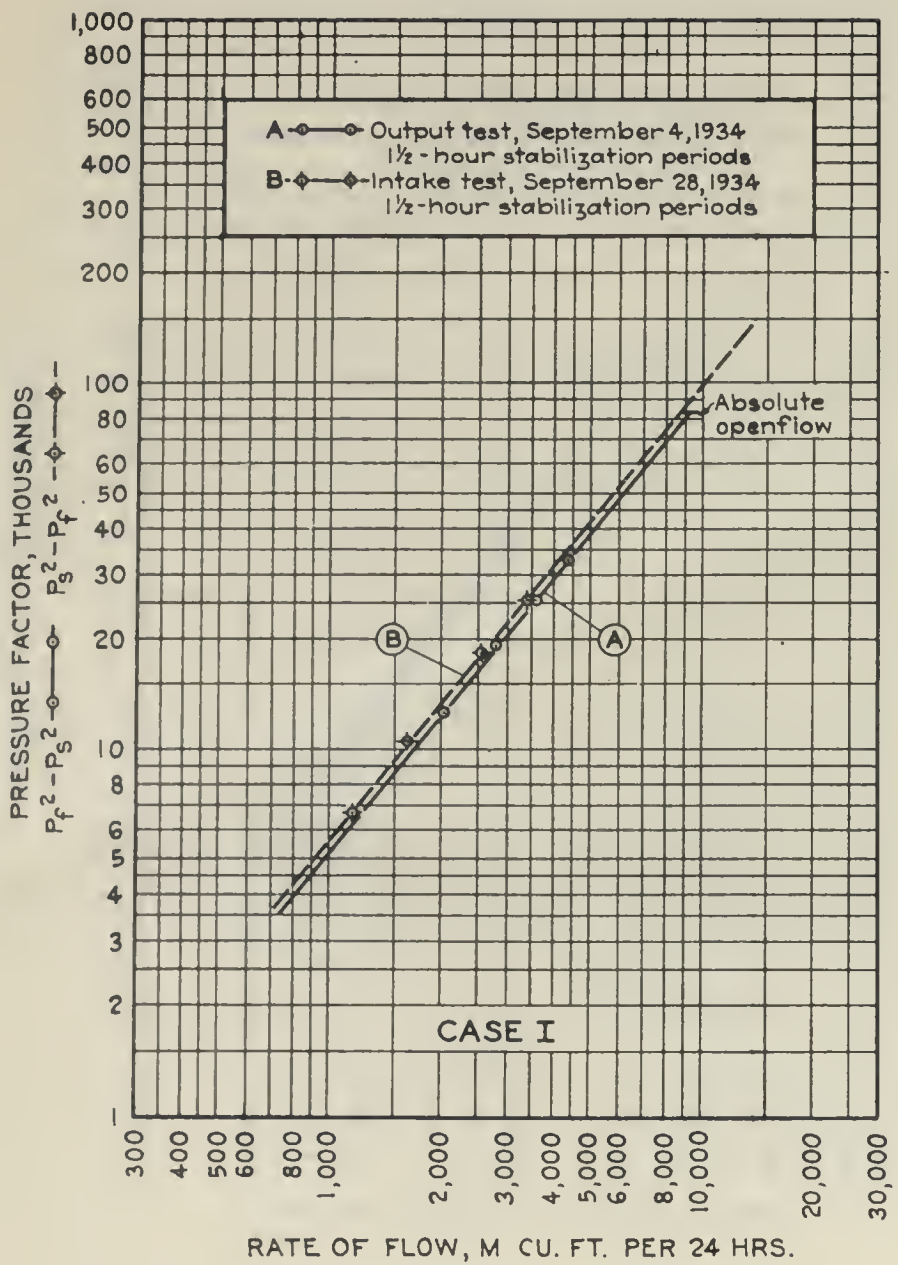

Intake test, September 6 to 12
1 September 6
2 " 7
3 " 8
4 September 9
5 " 10
6 " 11
7 September 12
A and B from case I, shown for comparison
CASE II
PRESSURE FACTOR, THOUSANDS
$P_f^2-P_s^2$ $P_s^2-P_f^2$
RATE OF FLOW, M CU. FT. PER 24 HRS.

Figure 7.—Comparison of output and intake characteristics of well B.



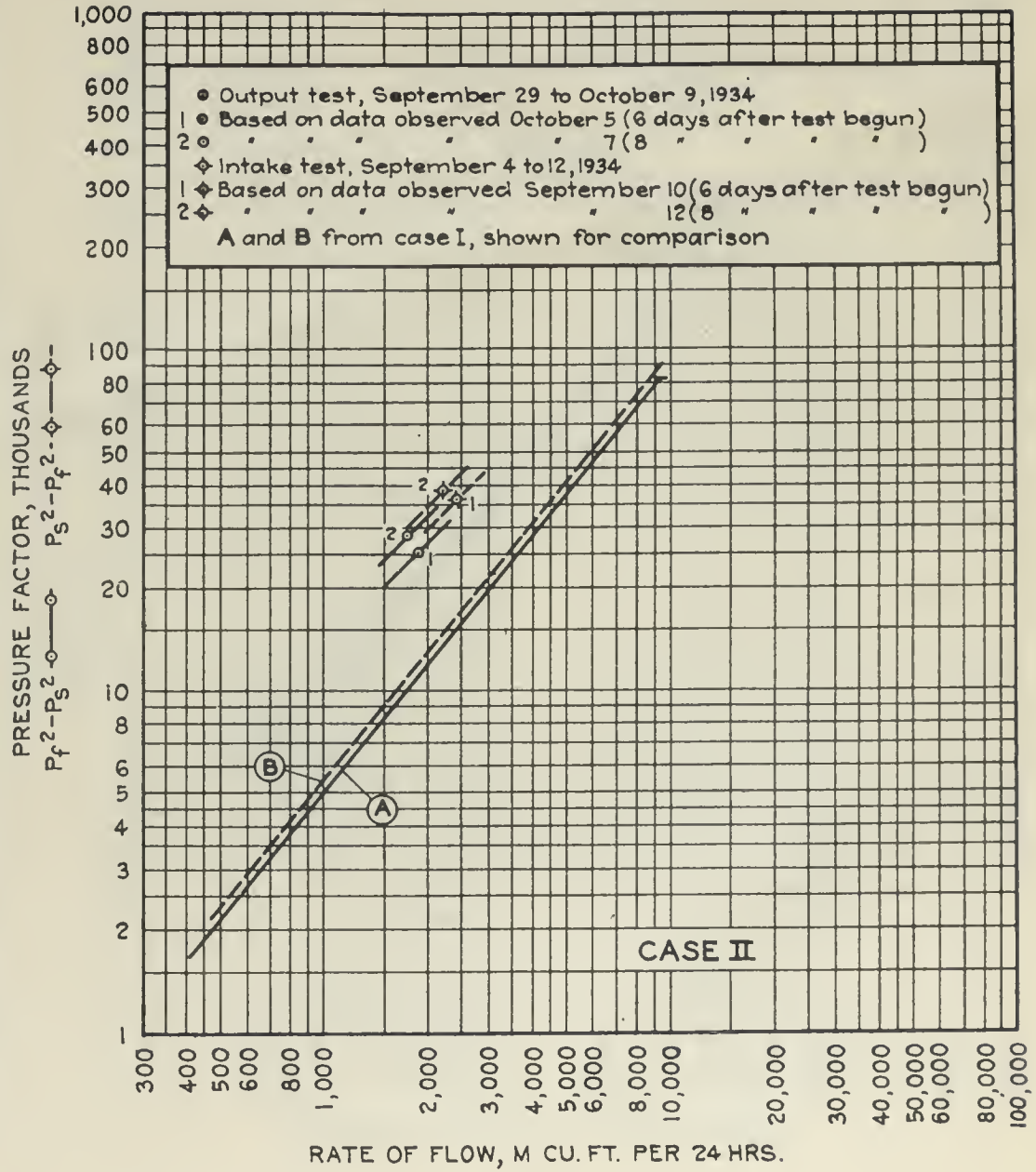
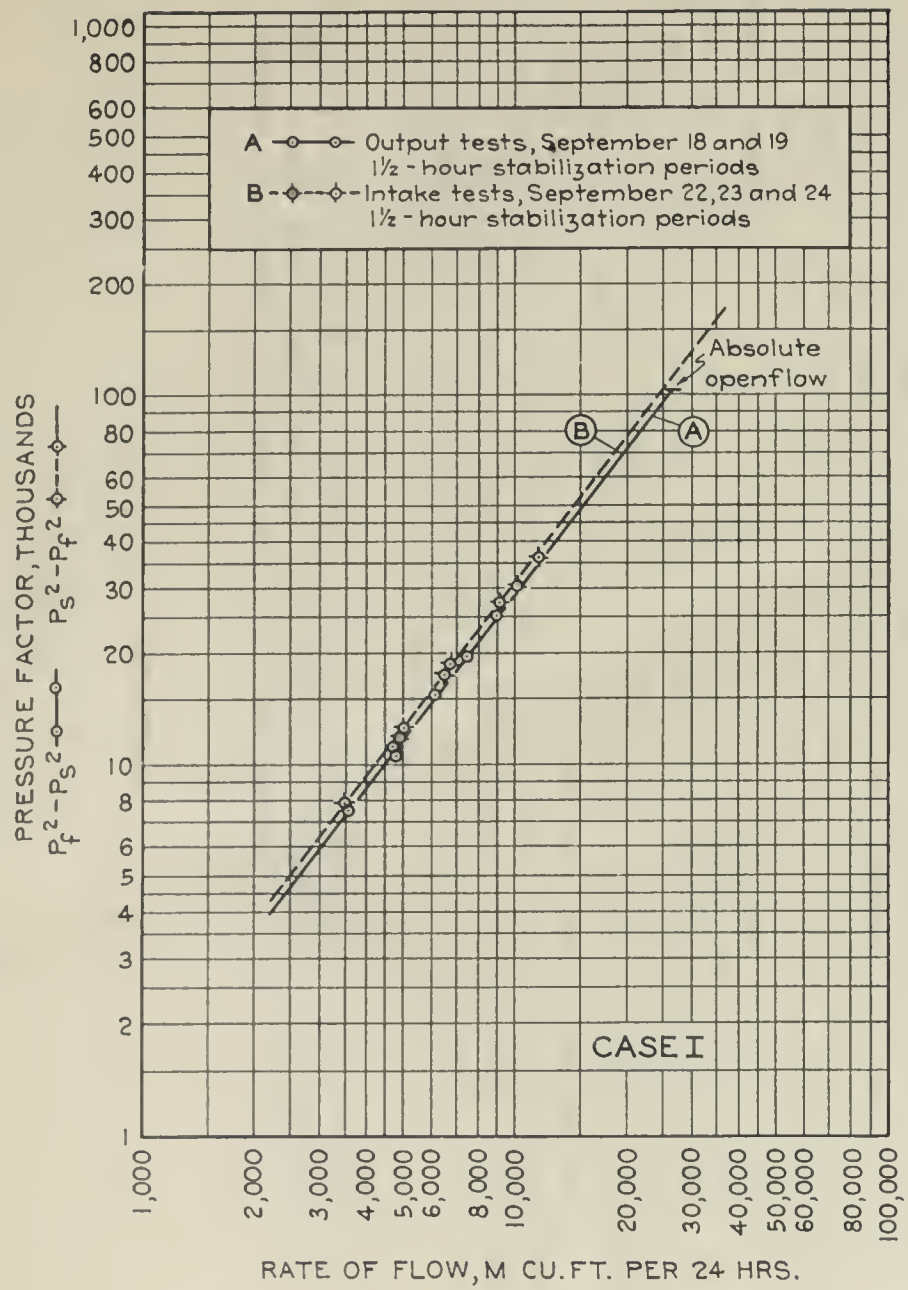

Output test, September 29 to October 9, 1934
1 Based on data observed October 5 (6 days after test begun)
2 " " " 7 (8 " " " ")
Intake test, September 4 to 12, 1934
1 Based on data observed September 10 (6 days after test begun)
2 " " " 12 (8 " " " ")
A and B from case I, shown for comparison
CASE II
PRESSURE FACTOR, THOUSANDS
$P_f^2-P_s^2$ — $P_s^2-P_f^2$
RATE OF FLOW, M CU. FT. PER 24 HRS.

Figure 8.—Comparison of output and intake characteristics of well C.



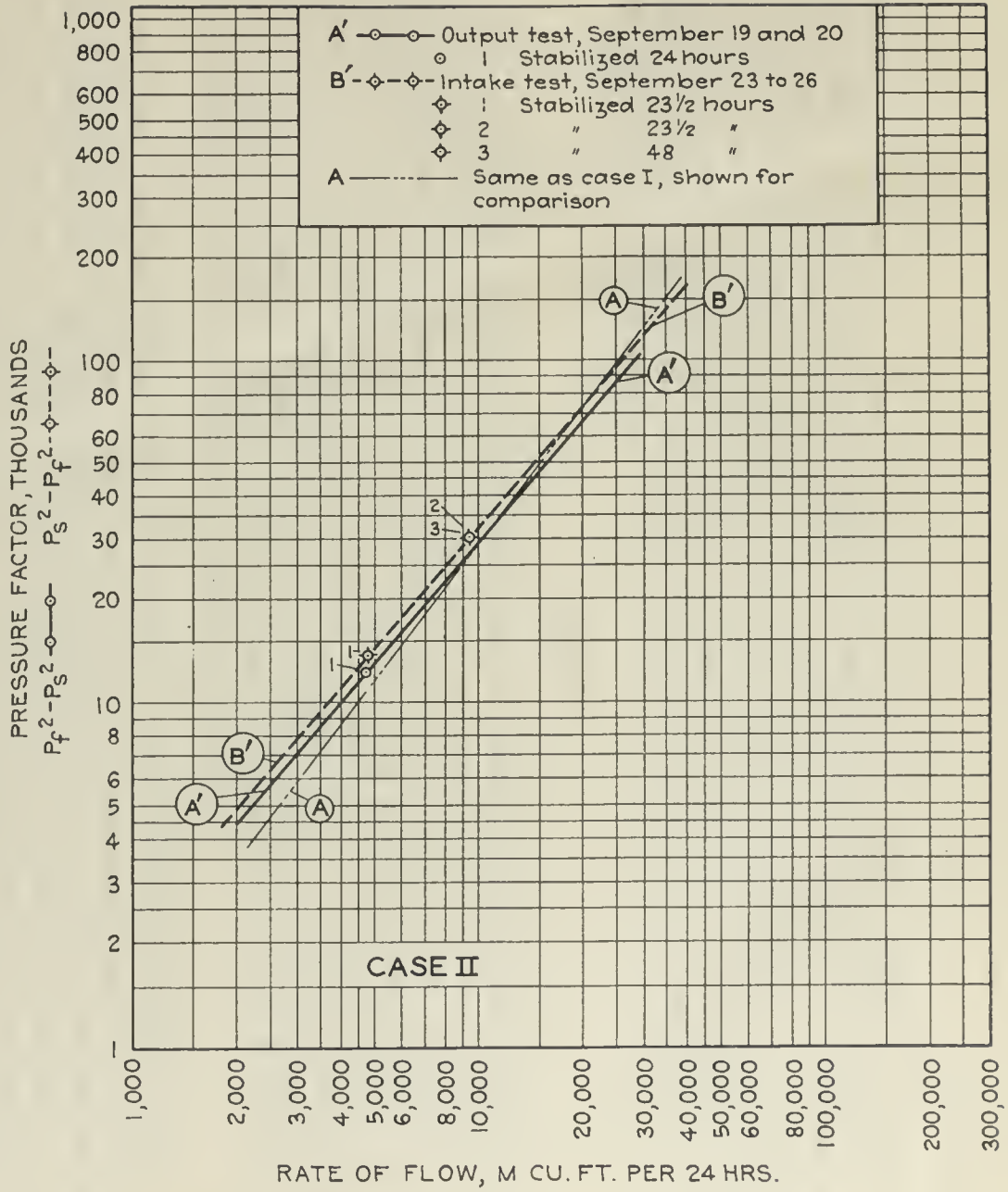


Figure 9.—Comparison of output and intake characteristics of well D.

continued. For example, the plotted data show that after outflow periods of 6 and 8 days, respectively, the rates of delivery from the well were only about 51 and 44 percent of the delivery rates at equal values of the pressure factor after stabilization periods of 1-1/2 hours. Similarly, it is shown that the rates of gas delivery into the well after 6 and 8 day periods, respectively, were only about 53 and 46 percent of the rates of delivery into the well at equal values of the pressure factor in the test employing stabilization periods of 1-1/2 hours.

Well D

The output and intake characteristics of well D, based upon stabilization periods of approximately 1-1/2 hours, are represented by lines A and B, respectively, case I, figure 9. The slope of the pressure-flow relationship plotted from the results of the inflow test, line B, is approximately equal to that of the pressure-flow relationship plotted from the results of the outflow test, line A. Comparison of the pressure-flow relationships represented by lines A and B, case I, shows that the rate of gas delivery from the well is about 6 percent greater than the rate of delivery into the well for the same values of the respective pressure factors.

The data plotted under case II, figure 9, were obtained under stabilized conditions of pressure and flow. Line B', representing the stabilized intake characteristic of the well, is based upon the results determined under the two delivery rates shown. Outflow data were obtained on the well for only one stabilized delivery rate; line A' was established by assuming that the slope of the line representing the outflow characteristic of the well was equal to that for the inflow characteristic represented by line B'. The rates of outflow indicated by line A' are about 7.5 percentgreater than the rates of inflow shown by line B', at equal values of the respective pressure factors. Comparison of the slope of the stabilized inflow pressure-flow relationship, line B', case II (fig. 9), with the slope of line B, case I, which was determined from data obtained after stabilization periods of 1-1/2 hours, shows that when the sequence in which the inflow data were obtained was such that the rate of delivery was increased in the series of operating conditions observed, tangent n of the angle between the pressure-factor (vertical) axis and the line representing the pressure-flow relationship increased as the pressure and flow became more nearly stabilized. This observation coincides with behavior noted during the study of the effect of stabilization on the producing characteristics of natural-gas wells.6/

6/ Rawlins, E. L., and Schellhardt, M. A., Back-Pressure Data on Natural-Gas Wells and Their Application to Production Practices: Monograph 7, Bureau of Mines ,(in coop. with Am. Gas Asso.), 1936, 210 pp.

Well E

Comparison of the results of the outflow and inflow tests on well E - lines A and B, respectively, figure 10 - indicates that the rate of gas delivery into the well was much less than the rate of delivery from the well. This difference might have been due to sealing of the face of the gas-producing sand by foreign materials, such as pipe scale, dust, or cavings, and to decreased permeability of the gas-producing formation caused by the infiltration of such materials into its pores when gas was being delivered into the sand. The relation of line C, which is based upon data obtained during the second outflow test, August 11, to lines A and B, illustrates the partial restoration of the delivery capacity of the well. Line D was determined from data obtained during the outflow test made August 12 after the well had been open-flowed; although the data for this test were somewhat erratic, the producing characteristic of the well as represented by line D shows a slightly increased delivery capacity compared with its producing characteristic represented by line A. Line E was determined from data obtained during the inflow test made on well E August 13. Comparison of D and E shows that there is relatively little difference between the slopes of the lines representing the outflow and inflow characteristics of the well and that throughout the range of data the rate of delivery of gas from the well was approximately 14 percent greater than the rate of delivery into the well for the same values of the respective pressure factors.

Well F

Line A, figure 11, is based upon the results of the outflow test made on well F August 15 and line B, upon data obtained during the inflow test made on the well August 16 and 17. Comparison of these lines shows an apparent difference in slopes; however, this difference is within the limits of the accuracy of the data generally obtained.

SUMMARY OF RESULTS

The results of the study of the output and intake characteristics of the natural-gas wells described in this report are summarized as follows:

1. The relationship established by plotting on logarithmic paper rates of gas delivery into the wells (Q) against corresponding values of the pressures at the face of the sand squared minus the formation pressure squared ($P_s^2 - P_f^2$) can be represented approximately by a straight line and mathematically by the equation.

$$Q = C\,(P_s^2 - P_f^2)^n,$$

where Q = rate of gas delivery into the well, cubic feet per 24 hours;

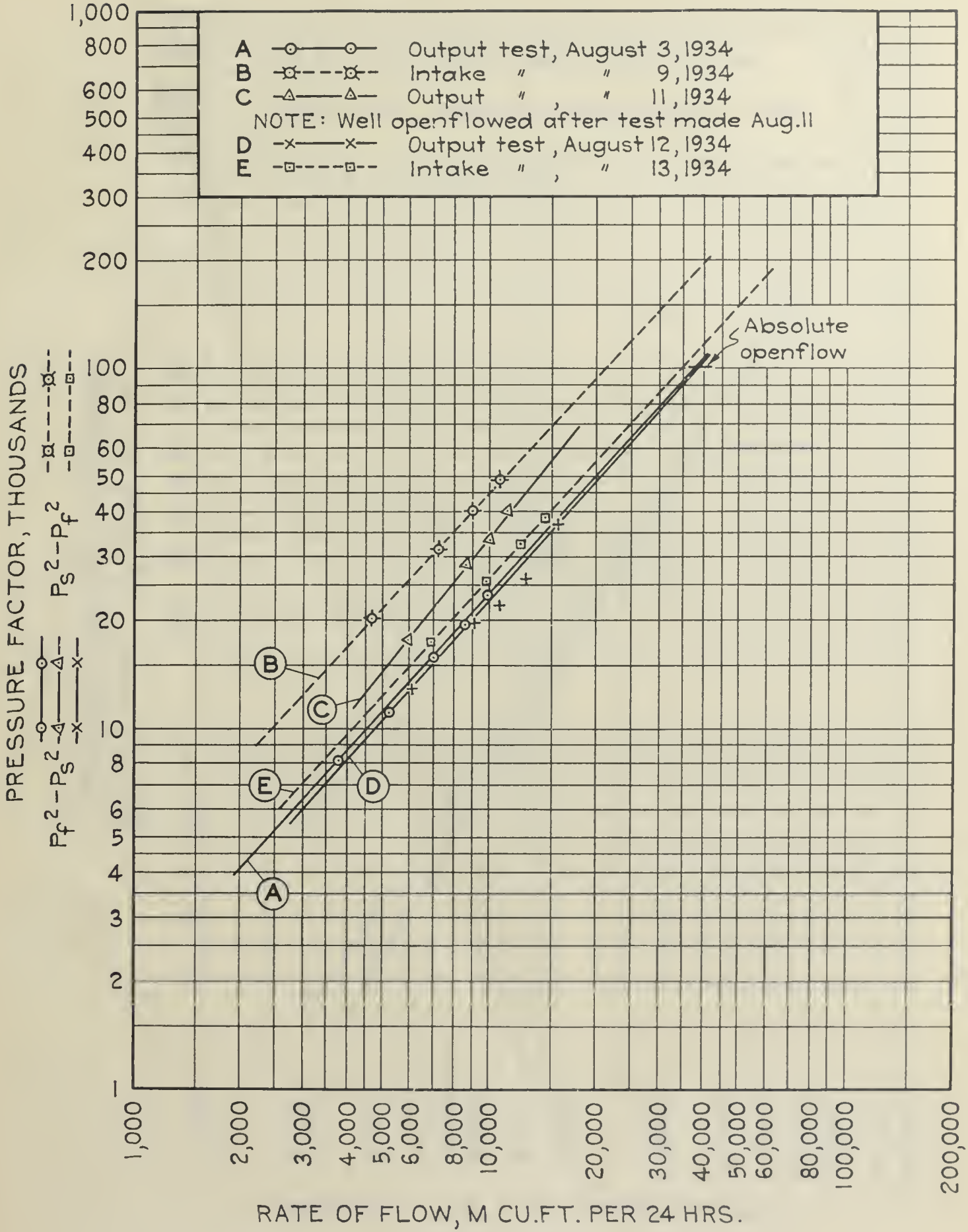


Figure 10.— Comparison of output and intake characteristics of well E.

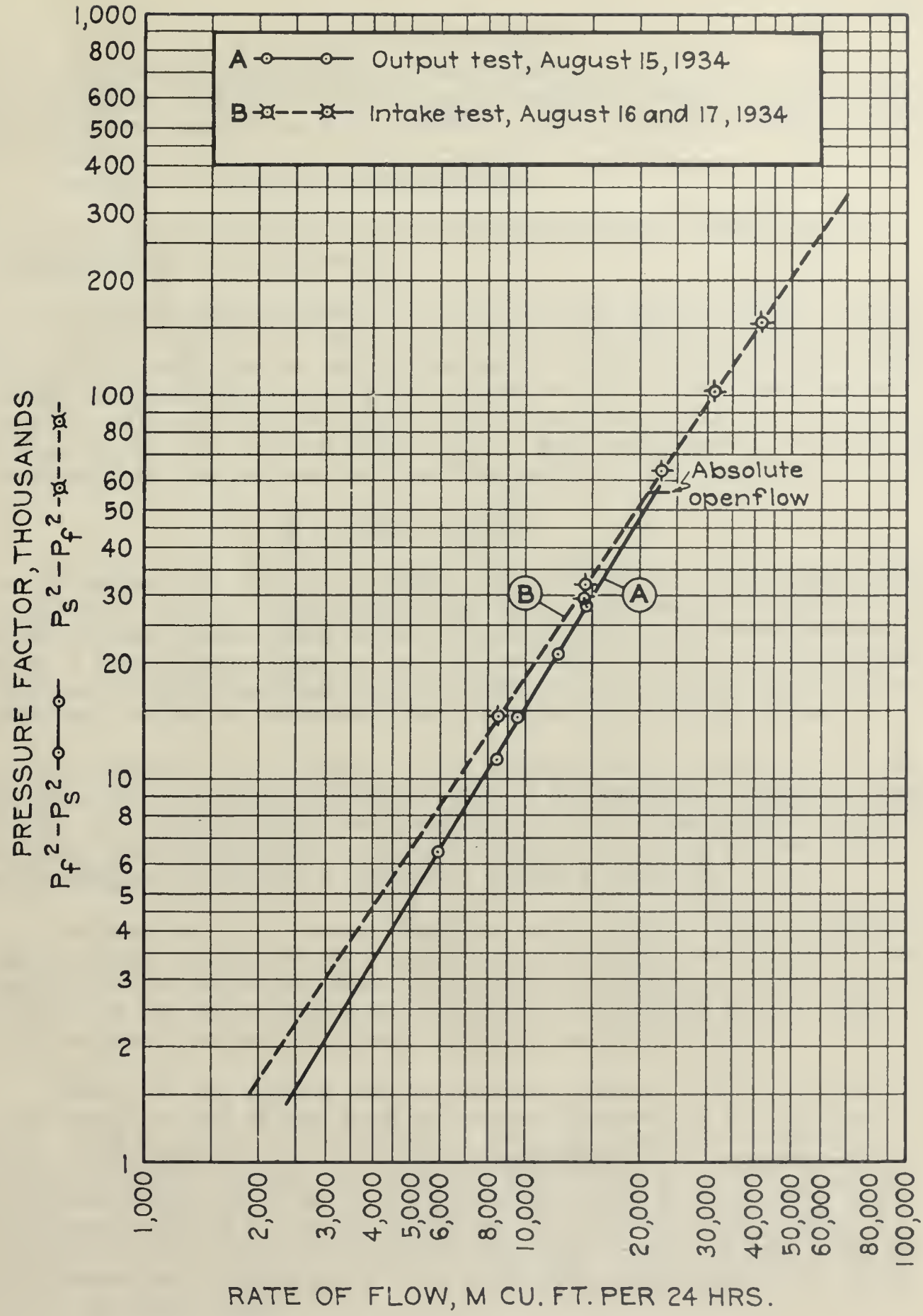


Figure 11.— Comparison of output and intake characteristics of well F.

C = coefficient, determined by experiment;
P_s = pressure imposed at face of sand, pounds per square inch absolute;
P_f = shut-in pressure at the sand face, pounds per square inch absolute; and
n = exponent, determined experimentally.

For the purpose of this report this relationship has been termed the "intake characteristic" of a gas well.

2. The intake characteristic of a gas well was determined in a manner similar to that originated in previous studies of output or producing characteristics of gas wells, where the pressure-flow relationship is established by plotting on logarithmic paper rates of gas delivery from a well against corresponding values of the formation pressure squared minus the back pressure at the face of the sand squared $(P_f^2 - P_s^2)$. This relationship is represented by a straight line and is expressed mathematically by the equation.

$$Q = C\ (P_f^2 - P_s^2)^n.$$

3. To compare the output and intake characteristics of the natural-gas wells described in this report both output and intake tests were made of each of the wells under similar conditions of pressure and flow stabilization, and rates of gas delivery from or into the well were plotted on logarithmic paper against the corresponding pressure factors $P_f^2 - P_s^2$ or $P_s^2 - P_f^2$ to determine the respective pressure-flow relationships (straight lines).

4. The slopes of the straight lines established by plotting rates of gas delivery from the respective wells against corresponding values of pressure factor $P_f^2 - P_s^2$ generally were approximately the same as the slopes of the straight lines established by plotting the rates of gas delivery into the wells against corresponding values of pressure factor $P_s^2 - P_f^2$.

5. The relation of the output delivery rate to the intake delivery rate under similar conditions of pressure and flow stabilization and at the same values of the respective pressure factors was such that rates of gas production from the wells generally were 4 to 7-1/2 percent greater than the rates at which gas was delivered into the respective wells.

6. Results of the outflow and inflow tests made on some of the wells show that coefficient C of the equation for the delivery of gas into gas wells, $Q = C\ (P_s^2 - P_f^2)$, will be subject to changes similar to those for coefficient C of the equation for flow from gas wells, $Q = C\ (P_f^2 - P_s^2)^n$. It is probable that liquids and foreign materials such as pipe scale and rock substances carried in the flow from source wells and from pipe-line facilities will affect the efficiency of operations seriously where gas is being delivered into a formation if these materials are permitted to enter the bore of an intake well.

7. Establishment of a pressure-flow relationship for the intake characteristic of a natural-gas well based upon the rate of gas delivery, Q, into the well and pressure factor $P_s^2 - P_f^2$ permits calculation of delivery rates that can be expected for different differential pressures $P_s - P_f$ through the formation into which gas is delivered, provided the shut-in formation pressure P_f is known.

8. Although the rates of delivery from and into a gas well at the same values of pressure factors $P_f^2 - P_s^2$ and $P_s^2 - P_f^2$ may be approximately the same, the delivery rates from and into a gas well at the same values of the differential pressure $P_f - P_s$ and $P_s - P_f$ are different. The following tabulation from the data obtained on wells A and B in this report compares rates of gas flow from and into wells under equivalent values of the differential pressure.

Differential pressure, $P_f - P_s$ or $P_s - P_f$, pounds per square inch	Well A[1]/			Well B[2]/		
	Rate of gas delivery, M cubic feet per 24 hours		Delivery into well in percentage of delivery from well	Rate of gas delivery, M cubic feet per 24 hours		Delivery into well in percentage of delivery from well
	From well	Into well		From well	Into well	
5	440	420	95.5	3,700	3,400	91.9
10	825	790	95.8	6,180	5,750	93.0
20	1,490	1,490	100.0	10,150	9,950	98.0
30	2,110	2,180	103.3	13,650	13,500	98.9
40	2,700	2,880	106.7	16,800	16,800	100.0
50	3,270	3,580	109.5	19,500	20,100	103.1
60	3,800	4,270	112.4	22,050	23,400	106.1
70	4,300	4,970	115.6	24,450	26,500	108.4
80	4,790	5,690	118.8	26,650	29,500	110.7
90	5,250	6,400	121.9	28,800	32,400	112.5
100	5,660	7,120	125.8	30,850	35,100	113.8

1/ Shut-in formation pressure, 330 pounds per square inch absolute. Delivery into well approximately 96 percent of delivery from well at equal values of pressure factors $P_s^2 - P_f^2$ and $P_f^2 - P_s^2$.

2/ Shut-in formation pressure, 341 pounds per square inch absolute. Delivery into well approximately 93 percent of delivery from well at equal values of pressure factors.

9. Results of the outflow and inflow tests conducted on the six gas wells in the Texas Panhandle field show a definite relationship throughout an appreciable range of operation between the output and intake characteristics of the relatively dry natural-gas wells in the Panhandle field from which the data were obtained; it is believed that intake characteristics calculated from the producing characteristics of natural-gas wells in this field will be accurate enough for projects where this information is needed.

10. Although the relation of output and intake capacities of gas wells was shown to be relatively independent of the stabilization characteristics of the wells, the results of the tests of well C emphasize the necessity for very thorough study of the producing characteristics of wells in order that input characteristics determined therefrom may be accurate.

After this report has served your purpose and if you have no further need for it, please return it to the Bureau of Mines. The use of this mailing label to do so will be official business and no postage stamps will be required.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

62.09
U
C p2

R. I. 3304

June 1936

Department of the Interior

United States Bureau of Mines
John W. Finch, Director

REPORT OF INVESTIGATIONS

PERMISSIBLE ELECTRIC MINE LAMPS

(SUPPLEMENT TO BULLETIN 332)

BY

L. C. Ilsley and A. B. Hooker

After this report has served your purpose and if you have no further need for it, please return it to the Bureau of Mines, using the official mailing label on the inside of the back cover.

R.I. 3304
June 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PERMISSIBLE ELECTRIC MINE LAMPS[1/]

(Supplement to Bulletin 332)

By L. C. Ilsley[2/] and A. B. Hocker[3/]

The last previous Bureau of Mines publication relative to details of permissible electric mine lamps is Bulletin 332, which describes 16 lamps approved prior to May 1930. Some of the lamps have become obsolete through the development of improved models which, with several new types, make it desirable to supplement the Bureau's published data relative to permissible electric mine lamps.

In the following descriptions, lamps approved since May 1930 and prior to July 1, 1935, are placed in the three following groups: (1) Electric cap lamps, (2) electric hand lamps, and (3) lamps for miscellaneous applications.

ELECTRIC CAP LAMPS APPROVED UNDER SCHEDULE 6B

1. Edison model J lamp, approval no. 24. - The features that distinguish this lamp from earlier models of Edison electric cap lamps are:

(a) The battery consists of two F-2 cells of square cross section and about 1-1/4 inches higher than H-14 cells in the model H lamp, making the battery narrower and higher than any of the earlier models of Edison cap lamps.

(b) The cells are insulated from each other and the Monel-metal casing by enclosure in vulcanized-rubber baglike enclosures. Both cell terminals are insulated from the cell jar.

(c) The battery is equipped with rubber-tube Bunsen-valve-type gas vents.

(d) The battery casing is secured by a new-type magnetic lock.

(e) The battery and cover terminal clips have been redesigned to give a wiping contact.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3304."

2/ Electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3/ Associate electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

(f) The lamp has approximately the same weight as the original Edison model C lamp approved in 1915 yet gives approximately three times as much light.

(g) The headpiece is the same as that used with the Edison model H lamp, except that it has a distinguishing green bezel ring.

(h) Three bulbs of different current rating are approved with the lamp, giving a choice of several time lengths of burning and corresponding amounts of light.

2. Edison model K lamp, approval no. 25. - This lamp differs from the model J lamp in the number of cells used, in weight, and in candlepower. The K lamp battery has three type F-2 cells and is therefore about 50 percent wider than that of the J lamp. The K lamp has approximately the same weight as the Edison model H cap lamp but owing to its higher voltage and greater bulb efficiency gives greater candlepower. It, too, has been approved with three bulbs of different current rating.

3. Super-Wheat model W lamp, approval no. 20. - This lamp differs from the earlier Super-Wheat model X lamp in the following points:

(a) The battery jar is thinner and taller.

(b) The complete battery is lighter and smaller due chiefly to the battery jar being made of lighter-weight rubber compound and having a more compact plate assembly, with little or no sludge space.

(c) The vent tube has been redesigned, and a simplified method of sealing the battery jars has been adopted.

(d) The battery plates are equipped with sponge-wood separators that greatly decrease the amount of free electrolyte in the battery jar.

(e) A single filler hole and plug serve both cells.

(f) The lamp is approved with a number of types of headpieces suitable for different methods of charging, but in every case the battery is charged through the headpiece. The cord enters the type V headpieces through the back, whereas in the type T headpieces it enters at the side.

The VA and TA headpieces are designed for the parallel system of charging in which the battery is automatically connected to the charging circuit by the action of a thermostatic switch in the headpiece. The VM and TM headpieces are for series charging in which the batteries are placed on charge by manual operation of a switch in the headpiece by means of a magnet.

(g) The headpiece is equipped with a main S-8 bulb and an auxiliary or emergency bulb. The main bulb has two major filaments, each having at least 100 hours' life. The emergency bulb is a small bulb placed beneath the

reflector but visible through a hole in the reflector. A switch at the side of the headpiece permits the connection at will of either filament of the main bulb or the emergency bulb.

Three main bulbs of different current rating are approved with the W lamp, giving a choice of several time lengths of burning and corresponding amounts of light.

4. Wheat model Q lamp, approval no. 17. - This lamp is the same as the model W Super-Wheat, except as to size, weight, and the amount of light that it supplies. Its battery casing is not as high or wide as that of the W lamp and is approximately 20 percent lighter. Two main bulbs of different current rating have been approved with the model Q lamp.

Further details of the cap lamps are given in table 1.

ELECTRIC HAND LAMPS APPROVED UNDER SCHEDULE 10B

1. Edison model H hand lamp, approval no. 1008H. - This lamp is a modification of the Edison model H electric cap lamp in which the headpiece is mounted directly on the battery cover, the cord being shortened to about 5 inches, just enough to reach through a special cover entrance to the regular cord terminals. The light is turned on and off by means of the switch on the headpiece.

The lamp is equipped with a suitable leather handle for carrying. The approval covers the use of either the regular diffusing-type reflector of the model H cap lamp or a palladium-plated brass reflector especially adapted for inspection service underground.

2. Edison model K hand lamp, approval no. 1008K. - This lamp is similar to the model H hand lamp except that it is a corresponding modification of the model K Edison permissible electric cap lamp; it has a 3-cell battery, making a lamp of greater weight but also of greater candlepower than the model H lamp.

3. Ecolite type 420BM lantern, approval no. 1012. - This is a railroad-type lantern modified for mine use by the addition of a safety device and a means of sealing the lens ring and battery-compartment cover. It is equipped with a 4-cell dry battery sufficient to supply a 0.15-ampere bulb for approximately eight 8-hour shifts or a 0.3-ampere bulb with its much greater light for three 8-hour shifts. The approval covers the optional use of either of these two bulbs.

The body of the lantern is rolled of sheet brass, cadmium-plated. A guard of bronze-alloy rod, cadmium-plated, attached as a shroud to protect the lens from breakage, also serves as a base or stool when the lantern is set down. An adjustable plated reflector permits focusing the light. This, with a large double convex lens results in a light beam of relatively high candlepower at the center for inspection purposes yet with enough side light for walking. A wooden handle adjustable to four positions permits carrying the lantern at various angles. The total weight of the lantern is 2-3/4 pounds.

Further details of the hand lamps are given in table 2.

TABLE 1. - Details of cap lamps

Lamp	Manufacturer	Approval no.	Date of approval	Kind of battery	No. of cells	Bulb symbol no.	Bulb rating	Bulb manufacturer	Approximate burning time per charge of battery with new lamps, hours	Total weight of lamp, pounds	Auxiliary of emergency light
Edison Model J	Thomas A. Edison, Inc.	24	6-22-31	Alkaline	2	BM-24	2.45 V. 1.0/0.75 A.	General Electric Co.	13	4.0	Minor filament in bulb
Do.	do.	24	11-2-32	do.	2	BM-24A	2.47 V. 0.9/0.65 A.	do.	14-1/4	4.0	Do.
Do.	do.	24	2-7-35	do.	2	BM-18H	2.4 V. 1.2/0.65 A.	do.	11	4.0	Do.
Edison Model K	do.	25	3-19-31	do.	3	BM-25	3.7/3.87 V. 1.0/0.65 A.	do.	13-1/2	5.4	Do.
Do.	do.	25	11-2-32	do.	3	BM-25A	3.75/3.87 V. 0.9/0.65 A.	do.	14-1/2	5.4	Do.
Do.	do.	25	2-7-35	do.	3	BM-25B	3.6/3.87 V. 1.2/0.65 A.	do.	10-1/2	5.4	Do.
Super-Wheat Model W	Koehler Mfg. Co., Inc.	20	8-16-33	Lead-acid Ironclad Exide	2	BM-20C	3.92 V. 1.0/1.0 A.	Westinghouse Lamp Co.	12	4.5	Separate bulb under reflector
Do.	do.	20	1-5-33	do.	2	BM-20B	3.98 V. 0.8/0.8 A.	do.	13-3/4	4.5	Do.
Do.	do.	20	7-11-33	do.	2	BM-20D	4.0 V. 0.67/0.67 A.	do.	20	4.5	Do.
Wheat Model Q	do.	17	8-2-34	do.	2	BM-17C	3.95 V. 0.55/0.55 A.	do.	15-1/2	3.6	Do.
Do.	do.	17	5-7-35	do.	2	BM-20D	4.0 V. 0.67/0.67 A.	do.	12-3/4	3.6	Do.

TABLE 2. - Details of hand lamps

Lamp	Manufacturer	Approval no.	Date of approval	Kind of battery	No. of cells	Bulb symbol no.	Bulb rating	Bulb manufacturer	Type of lock or seal	Total weight of lamp, pounds
Edison Model H	Mine Safety Appliances Co.	1008H	10-31-31	Edison Model H	2	BM-18H	2.4 V. 1.2/0.65 A.	General Electric Co.	Magnetic lock on battery casing; lead seal on light unit	5.0
Edison Model K	do.	1008K	10-31-31	Edison Model K	3	BM-25	3.7/3.87 V. 1.0/0.65 A.	do.	do.	5.3
						BM-25B	3.6/3.87 V. 1.2/0.65 A.			
Ecolite Model 42-BM	Economy Electric Co.	1012	10-17-33	Dry cells 1/	4	BM-3S	5 V. 0.15 A.	do.	Lead and wire seals	2.7
						BM-4S	5 V. 0.3 A.	Westinghouse Lamp Co. Tungsol Lamp Works		

1/ The approval covers the following lantern batteries:

Ray-O-Vac No. 941, manufactured by the French Battery Co.
Eveready No. 409, manufactured by the National Carbon Co.
Burgess No.1842, manufactured by the Burgess Battery Co.
Burgess No.1042, manufactured by the Burgess Battery Co.
Bright Star No. 460, manufactured by the Bright Star Battery Co.

LAMPS FOR MISCELLANEOUS APPLICATIONS APPROVED UNDER SCHEDULE 10-B

1. M.S.A. floodlight, models E and G, approval no. 1006. - This lamp consists of an automobile lighting unit, spotlight type, mounted on the casing of a portable battery of Edison cap-lamp cells, and a diffusing-type reflector that distributes the light over a 140° beam. The two models of this lamp are outwardly the same, differing only in battery and bulb. The model E lamp has a battery of six type E cells and a 6-8-volt bulb; the model G lamp has a battery of four type G cells and a 4.6-volt bulb of relatively higher current consumption.

The safety features consist of a bulb-ejection device, which pushes the bulb out of circuit when the bulb glass breaks, together with a snuffing or cooling action by the bulb guard which, when the bulb glass is broken, travels by strong spring action toward the bulb base, completes the breakage of the bulb glass, and mechanically cools the heated filament.

The battery casing is equipped with a padlock to prevent the hazard of exposed battery terminals; the headpiece is sealed with a lead and wire seal. The complete lamp weighs approximately 17 pounds.

2. Semiportable pneumatic-electric lamp, type L-18, approval no. 1007. - This lamp, of English manufacture, consists of an air turbine-magneto generator unit with an aluminum housing and an opal-glass dome around the electric bulb. The 6-pole rotor of the generator is made of cobalt steel and is mounted inside the moulded bakelite turbine wheel. A reducing valve lowers the air pressure of the line to 40 pounds as it enters the turbine. The lamp should therefore be connected to air pressures greater than 40 pounds.

The lamp is approved with a G.E. Mazda 1130, 21-cp., auto-type bulb. The opal-glass globe reduces the total light about 20 percent but gives a pleasing nonglare light.

The safety device is twofold: (1) In normal operation part of the turbine exhaust is bypassed through the space within the globe to maintain a pressure of about 2 pounds within the globe. This pressure prevents entrance of gas to the interior of the lamp. (2) If through breakage or leakage the pressure within the globe is reduced to near atmospheric, a special switch automatically short-circuits the generator terminals and thus extinguishes the bulb.

The lamp is designed for general lighting in gassy places wherever an adequate supply of compressed air is available. It may be hung at any desirable angle by an adjustable handle. The lamp is rugged and compact; its total weight is 10 pounds.

3. Edison type F_2 signal lamp, approval no. 1009. - This lamp consists of single Edison type F_2 cell with its rubber-bag container and a cylindrical battery jar or housing of steel pipe fitted with a screw-type cover that

holds the lighting unit - a 1.25-volt bulb and an ejection-type safety device enclosed in a ruby-glass dome that permits the signal light to be seen in all directions horizontally.

The housing is equipped with a hook for attaching the lamp either to the end or the side of a mine car.

The cover is locked to the housing by means of a magnetically operated lock. The total weight of the lamp is 7-1/2 pounds.

4. M.S.A. animal lamp, model E, approval no. 1010. - This lamp is made in two models, each consisting of a cap-lamp battery with a special light unit attached to the battery cover and equipped with a strap leather harness by which the lamp is suspended from the collar of the horse or mule.

The model E lamp uses the Edison model E battery and the same bulb, socket, and reflector assembly as the model E cap lamp but housed in a brass casting, which with an extra-heavy bezel ring prevents excessive damage to the lighting unit.

The model K lamp has an Edison model K battery and the complete model K headpiece unit including the bakelite shell with its regular bulb switch. Thus the light of the model K lamp may be turned off when not needed, especially when the horse or mule will not stand if left with his lamp lighted.

Models E and K are similar in appearance, the bakelite shell of the lighting unit being enclosed by a molded-brass casing like the brass shell of the model E unit.

The weight of each of these lamps is 6-1/3 pounds.

5. Wheat signal lamp, approval no. 1011. - This lamp, designed for trip lamp service, consists of a single Wheat lamp cell such as is used with the model K Wheat electric cap lamp but with soft rubber vulcanized to the hard rubber of the cell jar to make it cylindrical; the projecting base and side ribs, however, are of pliable rubber to resist the blows and bumps of service.

The lighting unit, comprising a 2-volt, 0.55-ampere bulb in a protective ruby-glass dome, is mounted over the battery and throws light in all directions horizontally. The lamp is fitted with a hook for clipping it over the end or side of a car and also with a bail for carrying it.

A switch at the side of the battery top permits the light to be turned off when it is not needed. This switch when moved to an extreme position by means of a magnet uncovers a charging receptacle through which the battery may be connected to a charging circuit by inserting a plug.

The total weight of the lamp is 6.4 pounds.

6. Justrite signal lamp, approval no. 1013. - The Justrite lamp has a battery of two flashlight (dry) cells and a small 0.15-ampere bulb, all mounted in a relatively heavy malleable-iron housing equipped with a handle at the top for carrying it and an adjustable hook attachment for hanging it on the end or side of a mine car. The bulb-enclosure part of the housing is equipped with a 3-inch-diameter ruby-glass lens at the front and three smaller 1-inch-diameter ruby lenses, one at each side and one at the rear. The red light is thus visible in all directions horizontally.

The battery of this lamp does not require charging or servicing; when the two cells have become nearly discharged it is only necessary to replace them with fresh ones.

The cover of the battery compartment is sealed with a wire and lead seal to prevent access to the battery except by an authorized person.

The total weight of the lamp is 11 pounds.

This lamp fills a need in haulage service at mines where facilities for charging portable-lamp batteries are not available, as at open-light mines.

7. M.S.A. special flashlight, approval no. 603. - This lamp is a modification of the Edison model K electric cap lamp for use by firebosses and other mine officials in mines where the use of electric cap lamps with flame safety lamps is prohibited, yet the use of flashlights is allowed for inspection purposes. To meet the requirements for this service, the Edison model K lamp was equipped with a push-button switch on the battery cover by which the cap lamp is lighted only while the switch is held in contact; this insures that the user will depend more upon his flame lamp than he would if the cap lamp were lighted continuously. In all other respects the lamp is identical with the Edison model K electric cap lamp, approval no. 25.

Further details of the miscellaneous lamps are given in table 3.

Table 3. - Details of miscellaneous lamps

Lamp	Manufacturer	Approval no.	Date of approval	Kind of battery	Battery manufacturer	No. of cells	Bulb symbol no.	Bulb rating	Bulb manufacturer	Type of lock or seal	Total weight of lamp, pounds
M.S.A. flood-light model E	Mine Safety Appliances Co.	1006E	2-11-31	Edison type M-14	Thomas A. Edison, Inc.	6	Mazda 88 BM-1006E	S-8 6-8 V. 1.69 A.	General Electric Co.	Padlock on battery casing; lead and wire seal on light unit	17-1/2
M.S.A. flood-light model G	do.	1006G	2-11-31	Edison type M-21	do.	4	BM-1006G	S-8 4.8 V. 2.5 A.	do.	do.	16-1/2
ML pneumatic-electric lamp	Joseph Lucas, Ltd., Birmingham, England.	1007	8-19-31	None (motor generator)	--		Mazda 1130	S-10 6-8 V. 2.36 A.	do.	Key lock	10
M.S.A. model F_2 signal lamp	Mine Safety Appliances Co.	1009	4-5-32	Edison type F_2	Thomas A. Edison, Inc.	1	BM-1-S	S-8 1.25 V. 0.9 A.	do.	Magnetic lock	7-1/2
M.S.A. model E animal lamp	do.	1010E	5-26-32	Edison type M-14	do.	2	BM-18-H	G-8 2.5 V. 1.1 A.	do.	do.	6-1/3
M.S.A. model K animal lamp	do.	1010K	10-25-35	Edison model K cap-lamp battery	do.	3	BM-25	3.70/3.87 V. 1.0/0.65 A.	do.	do.	6-1/3
Wheat signal lamp	Koehler Mfg. Co., Inc.	1011	5-26-32	Super-Wheat model X	Koehler Mfg. Co., Inc.	1	BM-2-S	S-8 2.0 V. 0.55 A.	Westinghouse Lamp Co.	Magnetically operated switch	6-2/5
Justrite signal lamp	Justrite Mfg. Co.	1013	3-21-34	(1)	(1)	2	BM-6-S	G-5 2.5 V. 0.15 A.	General Electric Co.	Lead and wire seal	11
Edison special lamp for use by mine officials	Thomas A. Edison, Inc.	603	8-14-34	Edison model K cap-lamp battery	Thomas A. Edison Inc.	3	BM-25B	S-8 3.6/3.87 V. 1.2/0.65 A.	do.	Magnetic lock	5-1/2

1/ The following two-cell flashlight batteries are approved with the Justrite lamp: Eveready No. 950, extra-long life, manufactured by the National Carbon Co., Inc. Burgess No. 2 cells, manufactured by the Burgess Battery Co. Ray-O-Vac No. 2 cells, manufactured by the Ray-O-Vac Co. New General Industrial Unit cells, manufactured by the General Dry Batteries, Inc.

CONCLUSIONS

This supplement to the published list of permissible electric mine lamps includes four new types:

1. A portable floodlamp for throwing a directed light on the working face.

2. A semiportable lamp for room lighting applicable wherever a suitable compressed-air supply is available.

3. An animal lamp for attachment to the collar of a horse or mule.

4. A dry-cell-type signal lamp especially applicable at mines where battery-charging facilities are not available.

The other lamps although not new types, represent considerable progress in mine-lamp design and manufacture. This progress includes the following:

1. Higher battery voltages by an increase in the number of cells.

2. Greater bulb wattage by an increase in battery voltage and bulb current.

3. Greater bulb efficiency through use of gas-filled bulbs and two filaments of approximately 100 hours' life instead of one filament of 200 hours' life.

4. More efficient use of battery plates.

5. Lighter-weight materials that reduce the total weight of the lamps.

6. A change in the Bureau's requirements for cap lamps to permit a more useful distribution of the extra light available in the later lamps and the optional use of higher-current bulbs that reduce the burning time per charge of battery to 10 hours.

7. More efficient reflectors.

The newer makes of cap lamps thus give several times as much useful light as the older models, with little or no additional weight. This is important to all concerned. To the miner it means better illumination per pound weight of lamp, more work accomplished, and greater safety through better visibility. To the mine operator it means greater efficiency, cleaner coal, and fewer accidents. To the lamp manufacturer it means increased sales through a wider use of his lamps.

Permissible electric mine lamps are more than safety lamps because they have been developed to the point where they compete successfully with open-flame lamps on the basis of efficiency. They give greater illumination than open-flame lamps; their light does not fluctuate with air movements or from variable fuel supply; they are not extinguished by sudden blasts and do not require attention during the working shift.

Direct competition with open-flame lamps in which the chief argument is based upon efficiency emphasizes the importance of illumination and encourages the design and sale of electric cap lamps of still greater light output while maintaining the present standards of safety and durability. From the user's standpoint the maximum desirable illumination seems to be limited only by the weight that can be comfortably carried and the possibility of excessive glare.

This trend toward greater illumination for mine workers possibly prophesies the day when electric cap lamps will be installed and maintained on the basis of guaranteed light output mutually agreed upon by the users and the manufacturers of the lamps.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

R. I. 3305 May 1936

DEPARTMENT OF THE INTERIOR
UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

INVESTIGATIONS DURING 1934 OF COMBUSTIBLES IN MANHOLES IN BOSTON, MASS.

This report presents the results of work done under a cooperative agreement between the Bureau of Mines, Department of the Interior, the Edison Electric Illuminating Company of Boston, and the Boston Consolidated Gas Co.

BY

G. W. JONES, JOHN CAMPBELL, F. M. GOODWIN,
AND W. P. YANT

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3305,
May 1936.

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR -- BUREAU OF MINES

INVESTIGATIONS DURING 1934 OF COMBUSTIBLES IN MANHOLES IN BOSTON, MASS.[1]

By G. W. Jones,[2] John Campbell,[3] F. M. Goodwin,[4] and W. P. Yant[5]

INTRODUCTION

The cooperative investigation dealing with the general hazards of combustible gases and vapors in manholes owned by the Edison Electric Illuminating Co. and the New England Telephone and Telegraph Co. in Boston, Mass., and surrounding territory, was continued during 1934.

The method and procedure of conducting the survey for 1934 were the same as those followed heretofore.[6]

ACKNOWLEDGMENTS

The cooperative investigation was arranged through the interest and effort of F. D. Comerford, president of the Edison Electric Illuminating Co. of Boston; E. M. Farnesworth, Jr., vice president and general manager of the Boston Consolidated Gas Co.; and A. C. Fieldner, chief engineer, Experiment Stations Division, U.S. Bureau of Mines. Thomas H. Haines, superintendent, Distribution Department, of the Edison Electric Illuminating Co., cooperated in furnishing both men and information for the work.

The New England Telephone and Telegraph Co., through Chief Engineer G. K. Manson, R. F. Esterbrook, H. G. Grush, and A. J. McCarron, cooperated by assigning a telephone company employee for the test truck. The company has borne its proportionate share of the expense and has taken an active interest in the work during the year. The test crew was operated under the direct supervision of John Campbell, superintendent of the Special Service Department of the Edison Electric Illuminating Co., and F. M. Goodwin, vice president of the Boston Consolidated Gas Co.

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3305."

2 Chemist, Gas Section, Pittsburgh Experiment Station, U.S. Bureau of Mines, Pittsburgh, Pa.

3 Superintendent, Special Service Department, Edison Electric Illuminating Co., Boston, Mass.

4 Vice President, Boston Consolidated Gas Co., Boston, Mass.

5 Supervising Engineer, Pittsburgh Experiment Station, U.S. Bureau of Mines, Pittsburgh, Pa.

6 Jones, G. W., and Perrott, G. St. J., Gases in Manholes: A Survey of a Public Utility in Boston, Mass.: Rept. of Investigations 3109, Bureau of Mines, 1931, 16 pp.

Jones, G. W., Investigations during 1931 of Gases in Manholes in Boston, Mass.: Rept. of Investigations 3192, Bureau of Mines, 1932, 32 pp.

Jones, G. W., Campbell, John, and Goodwin, F. M., Investigations during 1932 of Combustibles in Manholes in Boston, Mass.: Rept. of Investigations 3213, Bureau of Mines, 1933, 18 pp.

Jones, G. W., Campbell, John, and Goodwin, F. M , Investigations during 1933 of Combustibles in Manholes in Boston, Mass.: Rept. of Investigations 3260, Bureau of Mines, 1934, 26 pp.

A. H. Thomas, Distribution Department, Edison Electric Illuminating Co. of Boston, J. E. Sullivan, inspector of the Boston Consolidated Gas Co., and C. H. Wilder, of the New England Telephone and Telegraph Co., operated the test car during the survey. The testing was supervised by W. C. Johnson of the Special Service Department of the Edison Electric Illuminating Co.

W. E. Miller and E. S. Baker analyzed the samples of manhole atmospheres sent to Pittsburgh.

DISTRICT TESTED DURING 1934

The Edison Electric Illuminating Co. of Boston supplies electric power and light in Boston and 45 adjacent cities and towns. It has approximately 18,650 manholes in this territory; the majority are in Boston. The Boston Consolidated Gas Co. serves most of this territory with gas. The tests covered primarily the territories of the Edison Electric Illuminating Co. and the New England Telephone and Telegraph Co., irrespective of the distribution areas of the Boston Consolidated Gas Co.

SUMMARY OF 1934 RESULTS

Manholes of the Edison Electric Illuminating Co.

Table 1 summarizes the results obtained on the manholes of the Edison Electric Illuminating Co. during 1934. Of the 8,885 tests during the year, about 73 percent were made in Boston and the rest in the 24 towns listed in table 1.

The combustible gas concentration in 104 tests was 25 percent of the lower explosive limit, in 22 it was 50 percent, in 15 it was 75 percent, in 37 the atmospheres were explosive, and in 14 the combustibles present were above the upper explosive limit. In two tests there was a deficiency of oxygen and no appreciable quantity of combustibles.

Of the 7,720 manholes tested during the year, 159 (2.1 percent) showed the presence of combustibles, or a deficiency of oxygen.

Manholes of the New England Telephone and Telegraph Co.

The results of tests in manholes owned by the New England Telephone and Telegraph Co. are summarized in table 2. The manholes of this company were in the same towns and usually in the same streets as those of the Edison Electric Illuminating Co. Of the 3,572 tests during the year about 75 percent were made in Boston and the rest in the 20 towns listed in table 2.

The combustible gas concentration in 178 tests was 25 percent of the lower explosive limit, in 58 it was 50 percent, in 22 it was 75 percent, in 39 the atmospheres were explosive, and in 19 the combustibles present were above the upper explosive limit. In 46 tests there was a deficiency of oxygen. Of the 2,907 manholes tested, 212 (7.3 percent) showed combustibles or a deficiency of oxygen during the year.

The results obtained in the manholes of the Edison Electric Illuminating Co. from 1929 to 1934, and those obtained in manholes of the New England Telephone and Telegraph Co. for 1933 and 1934 are shown graphically in figure 1.

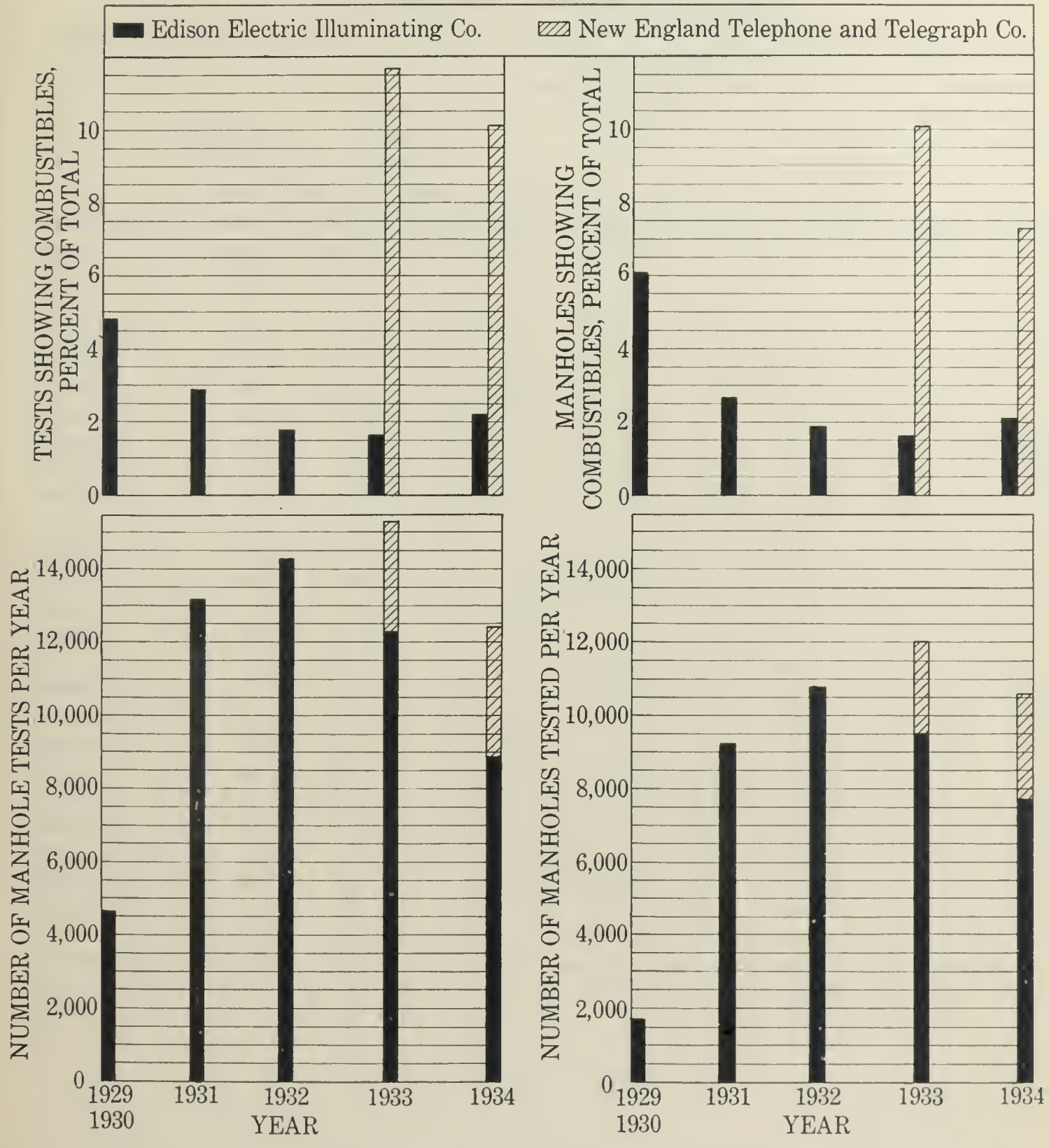


Figure 1.- Comparison of 1934 results with previous surveys.

TABLE 1.- Summary of tests in manholes of the Edison Electric Illuminating Co. in the Boston district during 1931

Town	Total number of manhole tests made	Number of manholes tested	Average number of tests per manhole per year	Combustibles found: Relation to explosive range: Above	Explosive	75 percent	50 percent	25 percent	10 percent or less	"Soil" gas-low oxygen content	Manholes showing combustibles during the year: Number	Percent of number tested
Boston	6,462	5,443	1.2	12	30	14	20	86	6,298	2	133	2.4
Brookline	441	391	1.1	0				0	441			.0
Canton	18	18	1.0						18			.0
Charlestown	18	18	1.0						18			.0
Chelsea	135	122	1.1		1	1	1	2	130		4	3.3
Concord	52	52	1.0						52			.0
Dedham	103	90	1.1					1	102		1	1.1
Framingham	70	67	1.1					5	65		3	4.5
Lexington	67	67	1.0						67			.0
Milton	41	41	1.0						41			.0
Natick	76	76	1.0						76			.0
Newton	313	285	1.1		3		1	4	305	0	8	2.8
Norwood	16	16	1.0						16			.0
Quincy	117	109	1.1						117			1.0
Reading	15	15	1.0						15			.0
Somerville	404	397	1.0					2	402		2	.5
Stoneham	44	39	1.1		1				43		1	2.6
Wakefield	32	27	1.2	2	1				29		3	10.3
Walpole	28	28	1.0						28			.0
Waltham	120	115	1.0		1				119		1	.9
Watertown	131	127	1.0						131			.0
Wellesley	26	26	1.0					1	25		1	3.8
Westwood	18	18	1.0						18			.0
Winchester	60	56	1.1					3	57		2	5.3
Woburn	78	77	1.0						78			.0
Total	8,895	7,720	1.2	14	37	15	22	104	8,691	2	159	2.1

TABLE 2.- Summary of tests in manholes of the New England Telephone & Telegraph Co. in the Boston district during 1934

Town	Total number of manhole tests made	Number of manholes tested	Average number of tests per manhole per year	Combustibles found: Relation to explosive range: Above	Explosive	75 percent	50 percent	25 percent	10 percent or less	"Soil" gas-low oxygen content	Manholes showing combustibles during the year: Number	Percent of number tested
Boston	2,657	2,072	1.3	17	33	19	53	148	2,341	46	175	8.4
Brookline	215	192	1.2		2	2	5	16	220		18	9.4
Canton	5	5	1.0						5			.0
Charlestown	10	10	1.0						10			.0
Chelsea	64	60	1.1					1	63		1	1.6
Concord	12	12	1.0						12			.0
Dedham	5	5	1.0						5			.0
Framingham	27	26	1.0						27			.0
Lexington	20	14	1.0					2	18		2	14.3
Natick	28	27	1.0					2	26		1	3.7
Newton	115	111	1.0					2	113		2	1.8
Quincy	11	11	1.0						11			.0
Somerville	195	191	1.0	2				2	191		4	2.1
Stoneham	13	13	1.0						13			.0
Wakefield	6	5	1.2			1			5		1	20.0
Walpole	3	3	1.0						3		0	.0
Waltham	41	41	1.0						41			.0
Watertown	55	49	1.1		1			2	50		3	3.0
Wellesley	21	21	1.0					2	19		2	9.5
Winchester	19	19	1.0					1	18		1	5.3
Woburn	22	20	1.1		3				19		2	10.0
Total	3,572	2,907	1.2	19	39	22	58	178	3,210	46	212	7.3

The steady decrease in the percentage of manhole tests showing combustibles from the beginning of the survey in 1929 until the end of 1933 is significant. For 1934 the percentage of manhole tests containing combustibles increased slightly, from 1.6 to 2.2 percent. This increase was due to the extremely cold weather in Boston in February 1934. Forty-eight percent of the manhole tests for the entire year that showed combustibles present were made during February and March; the presence of combustibles at that time was a result of the abnormally cold, freezing weather. Extensive periods of such weather cause frost to penetrate farther into the soil, and gas mains are thus subjected to greater shock hazards from trucks and other heavy vehicles passing over the street surfaces. This increased shock hazard, along with the contraction of the gas mains caused by the lowered soil temperatures, produces a greater number of gas leaks. In spite of the severe weather conditions during this period the percentage for the year is only slightly higher than for the 1932 and 1933 surveys and lower than the 1929-30 and 1931 surveys.

When manholes containing combustibles are compared, the 1934 survey showed that 2.1 percent of the manholes tested contained combustibles at one time or another during the year. This value is also slightly above the values of 1.9 and 1.6 percent for 1932 and 1933, respectively, but much below the values of 6.1 and 2.7 for 1929-30 and 1931.

The New England Telephone and Telegraph manhole survey for 1934 showed improvement over the 1933 survey. The percentage of tests indicating combustibles or a deficiency of oxygen was 10.1 percent for 1934 compared with 11.7 for 1933. When compared on the basis of the number of manholes showing combustibles to the total number tested, there were 7.3 percent during 1934 and 10.1 percent during 1933.

The percentage of manholes, and manhole tests, showing combustibles is higher (as shown in fig. 1) for the New England Telephone and Telegraph Co. than for the Edison Electric Illuminating Co. manholes. This is attributed to the fact that most of the telephone manholes have nonventilated covers, which thus reduces the amount of ventilation in these manholes and prevents combustibles from escaping to the air above.

If the tests made in the Edison Electric Illuminating Co. and New England Telephone and Telegraph Co. manholes are considered as a whole, the average results for all tests show a decrease in the combustible hazards in the manholes for 1934 compared with the previous year.

Gas Leaks Repaired as a Result of the Cooperative Survey

Table 3 gives the number of leaks repaired by the Boston Consolidated Gas Co. during 1934 in consequence of the manhole investigation. As a direct result of the cooperative tests in Edison Electric Illuminating Co. manholes, 132 repairs were made, and tests in manholes of the New England Telephone and Telegraph Co. resulted in 207 repairs, making 339 in all.

Other gas companies besides the Boston Consolidated Gas Co. operate in the territory where tests were conducted. When combustibles were found in manholes in these territories the responsible companies were notified and the conditions usually remedied. Data on the number of leaks repaired by these outside companies are not available.

As in previous years, the test crew reports the presence of combustibles in manholes to all companies concerned at frequent intervals during the day. The gas company immediately sends one of its "cruising cars" to the location given, and the gas mains in the particular district are inspected. If leaks are found repairs are made immediately.

After about 2 weeks (usually the Saturday following the 2-week period) the test crew retests the manholes that have shown combustibles to determine whether the repairs made were effective in eliminating the hazards. If combustibles are still present a reinspection is made by the gas company. This process is continued until either the contamination is eliminated or the trouble is found to be due to other causes.

Composition of Manhole Atmospheres

A number of representative manhole atmospheres taken during 1934 in Edison Electric Illuminating Co. manholes that contained combustibles or were found deficient in oxygen are given in appendix A. Appendix B lists the composition of representative manhole atmospheres of the New England Telephone and Telegraph Co. The method of sampling and the procedure used in the analysis of manhole atmospheres have been described in previous reports.[6]

Table 4 states the range in composition of the constituents in the manhole atmospheres of the Edison Electric Illuminating Co. and New England Telephone and Telegraph Co. manholes during 1934.

TABLE 3.- Leaks repaired by the Boston Consolidated Gas Co. during 1934 as a result of manhole investigation

Month	January		February		March		April		May		June		July		August	
Utility manholes tested	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N.E.T. & T.
Manhole tests made	956	313	511	53	782	401	893	337	819	387	550	387	814	365	1124	236
Samples showing combustibles by detector	10	22	43	11	48	52	7	15	14	21	19	75	13	42	8	5
Leaking joints repaired	0	13	3	1	13	13	1	2	6	4	10	45	12	28	3	0
Broken mains repaired	1	0	26	5	15	6	0	1	0	0	0	2	0	2	0	0
Leaking and broken services repaired	4	2	2	4	5	2	2	5	4	3	1	3	0	0	0	0
Miscellaneous repairs[1]	0	0	0	0	2	6	0	1	0	0	2	1	0	0	0	0
Total leaks repaired	5	15	31	10	35	27	3	9	10	7	13	51	12	30	3	0

Month	September		October		November		December		Total		
Utility manholes tested	Edison	N.E.T. & T.	Edison	N.E.T. & T.	Edison	N E.T. & T.	Edison	N.E.T. & T.	Edison	N E.T. & T.	Total
Manhole tests made	684	248	579	332	603	250	570	263	8,885	3,572	12,457
Samples showing combustibles by detector	8	16	8	29	4	27	7	12	189	327	516
Leaking joints repaired	3	7	2	15	2	9	0	5	55	142	197
Broken mains repaired	0	0	0	2	1	4	7	0	50	22	72
Leaking and broken services repaired	2	1	11	0	1	11	0	1	22	32	54
Miscellaneous repairs[1]	0	0	1	1	0	1	0	1	5	11	16
Total leaks repaired	5	8	14	18	4	25	7	7	132	207	339

1/ Includes repairs to broken gates, sand holes in pipe, drip stems, test pipes, and split sleeves.

TABLE 4.- Range in composition of manhole atmospheres, 1934

Constituents	Edison Electric Illuminating Co. manholes, percent by volume	New England Telephone & Telegraph Co. manholes, percent by volume
Carbon dioxide	0 to 2.0	0 to 9.0
Illuminants	0 to 2.3	0 to 1.7
Oxygen	12.8 to 20.9	3.5 to 20.9
Hydrogen	0 to 15.1	0 to 23.9
Carbon monoxide	0 to 6.8	0 to 2.6
Methane-ethane	0 to 9.9	0 to 12.5
Gasoline vapor	0 to .3	0
Nitrogen	57.4 to 80.0	45.8 to 81.7

Table 5 reviews the results for previous surveys, while Table 6 states the maximum range for the 6-year period, 1929-34.

TABLE 5.- Range in composition of manhole atmospheres, 1929-33, percent by volume

Constituents	1929-1930 survey	1931 survey	1932 survey	1933 survey
Carbon dioxide	0 to 7.7	0 to 16.6	0 to 8.3	0 to 16.7
Illuminants	0 to 2.3	0 to 3.9	0 to 2.5	0 to 1.6
Oxygen	7.4 to 20.9	1.0 to 20.5	3.7 to 20.9	3.3 to 20.9
Hydrogen	0 to 25.3	0 to 20.5	0 to 44.1	0 to 15.8
Carbon monoxide	0 to 5.1	0 to 3.8	0 to 3.4	0 to 2.1
Methane-ethane	0 to 22.2	0.1 to 34.7	0.1 to 25.4	0 to 13.4
Gasoline vapor	0	0 to 0.6	0 to 0.1	0
Nitrogen	43.2 to 81.5	30.6 to 83.7	19.5 to 85.6	57.0 to 83.0

TABLE 6.- Range in composition of manhole atmospheres, 1929-34, percent by volume

Constituents	Maximum range
Carbon dioxide	0 to 16.7
Illuminants	0 to 3.9
Oxygen	1.0 to 20.9
Hydrogen	0 to 44.1
Carbon monoxide	0 to 6.8
Methane-ethane	0 to 34.7
Gasoline vapor	0 to .6
Nitrogen	19.5 to 85.6

SOURCE OF GASES IN MANHOLES

A study of the constituents found in the manholes in Boston and contiguous territory during the 6-year survey shows that the contaminations may be divided into three general classes, as follows:

1. Gases containing hydrogen, carbon monoxide, illuminants, methane, and ethane. These contaminations have been found to be due usually to leakage of manufactured gas from gas mains and services. At times gases of a similar nature may have been produced by thermal decomposition of hydrocarbon insulation materials in the manhole or duct lines.

2. Gases containing high percentages of carbon dioxide, varying quantities of methane, little or no carbon monoxide, hydrogen, illuminants, and less than 17 percent oxygen. Most of these mixtures were nonexplosive. Atmospheres of this type have been termed "soil" gas because the low oxygen content appears to have been caused by reaction between the oxygen in the air and organic or other materials in the soil. The small percentages of carbon monoxide, hydrogen, and illuminants that are often present in the manhole atmospheres coming under the general classification of "soil" gases may be due to the leakage of small amounts of manufactured gas into the manholes or to bacterial reactions of organic materials in the soils adjacent to the manholes.

3. Gases containing gasoline vapor. In the atmospheres sampled the amount of gasoline vapor present was insufficient to produce explosions.

Contaminations from more than one source were found in many of the samples analyzed, and often there was no sharp dividing line between the classes of contamination.

SUMMARY

A survey of explosion hazards in manholes owned by the Edison Electric Illuminating Co. in Boston and adjoining towns and in manholes of the New England Telephone and Telegraph Co. in the same territory was continued during 1934. In all, 8,885 tests were made in manholes of the Edison Electric Illuminating Co., and 3,572 tests in manholes of the New England Telephone and Telegraph Co.

Results obtained in manholes of the Edison Electric Illuminating Co. showed a slight increase, while those of the New England Telephone and Telegraph Co. showed a decrease in explosion hazards for 1934 when compared with 1933. Tests made in manholes of both companies, when considered together, showed a decrease in explosion hazards for 1934 when compared with 1933, both on the basis of total number of manhole tests and number of manholes tested.

A general survey of the constituents present in manholes during the year showed three general types of contamination: (1) Gases containing hydrogen, carbon monoxide, illuminants, methane, and ethane, usually due to manufactured gas that had found its way into manholes from leaks in gas mains and services; (2) gases containing abnormally high percentages of carbon dioxide, varying amounts of methane, and low percentages of oxygen, termed "soil" gases; and (3) gasoline vapor.

Contamination from more than one source was not uncommon, and often there was no sharp dividing line between the different classes of contamination.

The 1934 survey again has shown clearly the value and necessity of cooperative, systematic, and continuous manhole testing to maintain the explosion hazards in such underground openings at a minimum. The cooperative investigation is being carried on during 1935 and 1936.

APPENDIX A

Analyses of manhole atmospheres, Boston, 1934

(Representative manhole atmospheres of the Edison Electric Illuminating Co.)

Manhole No.	Town	Date, 1934.	Composition, percent by volume								Calculated explosibility	Explosibility by detector		Weather
			CO_2	Illuminants	O_2	H_2	CO	$CH_4 + C_2H_6$	Gasoline + other hydrocarbons	N_2		Without filter	With filter	
1	Boston	3-2	0.2	0.2	19.8	0.8	0.2	0.3	0.0	78.5	25	25	25	Clear.
2	Do.	2-21	.2	.5	18.7	4.9	.5	2.8	.0	72.4	100	100	100	Do.
3	Do.	2-21	.4	.8	16.3	10.4	1.0	5.9	.0	65.2	.....	Above	100	Do.
6	Do.	2-21	.1	.3	20.0	1.5	.2	.7	.0	77.2	45	25	25	Do.
10	Do.	3-5	.4	.0	19.1	.4	Trace	.6	.0	79.5	15	25	25	Do.
11	Do.	3-5	.3	.3	18.8	4.3	.4	2.4	.0	73.5	100	100	100	Do.
14	Do.	3-5	.7	.2	16.1	.8	.1	2.1	.0	80.0	55	75	75	Do.
17	Do.	8-31	.5	.3	19.9	.0	.0	1.1	.0	78.2	25	25	25	Do.
18	Do.	6-11	.3	.1	20.1	.5	.0	.8	.0	78.2	25	25	25	Do.
19	Do.	12-20	.2	.2	19.6	2.4	.4	1.3	.0	75.9	70	75	75	Do.
20	Do.	12-20	.3	.4	19.1	3.8	.3	2.1	.0	74.0	100	100	100	Do.
22	Do.	12-20	.2	.2	19.6	2.4	.3	1.3	.0	76.0	70	75	75	Do.
25	Do.	5-17	.3	.3	19.0	2.7	.2	2.0	.0	75.5	85	100	75	Do.
25	Do.	6-1	.6	.3	18.6	2.9	.2	2.1	.0	75.3	90	100	100	Do.
29	Do.	2-12	.3	.4	18.2	5.0	.9	2.7	.0	72.5	100	100	100	Do.
30	Do.	2-15	.2	.2	19.6	2.3	.4	1.0	.0	76.3	65	75	50	Do.
33	Do.	3-6	.3	.5	12.8	5.4	.6	5.5	.0	74.9	.....	Above	100	Do.
34	Do.	2-23	.2	.3	18.8	3.5	.4	1.8	.0	75.0	100	100	100	Do.
35	Do.	2-23	.4	.8	16.1	11.0	1.0	6.6	.0	64.1	.....	Above	100	Do.
37	Do.	7-5	.2	.0	20.3	.5	.1	.6	.0	78.3	20	25	25	Do.
40	Do.	7-24	1.0	.2	18.1	.0	.0	1.2	.0	79.5	25	25	25	Do.
41	Do.	11-22	.6	.2	17.5	1.3	.1	2.5	.0	77.8	70	100	75	Do.
44	Do.	6-11	.8	.7	17.3	6.3	.7	4.1	.0	70.1	100	100	100	Do.
45	Do.	2-15	.2	.1	19.7	2.2	.3	.8	.0	76.7	55	75	50	Cloudy.
47	Do.	2-7	.4	.2	18.3	1.1	.2	1.0	.0	78.8	40	50	25	Clear.
49	Do.	5-15	.4	.3	18.7	1.4	.1	1.7	.0	77.4	60	75	75	Rain.
50	Do.	12-27	.2	.3	19.5	3.0	.2	1.6	.0	75.2	85	75	75	Clear.
53	Do.	3-13	1.0	1.0	13.9	3.8	.5	8.0	.0	71.8	.....	Above	100	Do.
55	Do.	3-13	.2	.3	18.4	1.6	.2	1.6	.0	77.7	60	75	50	Do.
62	Do.	3-28	.1	.1	18.8	.3	Trace	1.1	.0	79.6	25	25	25	Rain.
64	Do.	5-17	.5	.4	17.7	5.5	.1	3.4	.0	72.4	100	100	100	Clear.
65	Do.	2-6	.9	.2	17.1	1.2	.2	2.7	.0	77.7	70	75	75	Do.
70	Do.	2-28	.3	.7	17.8	5.2	1.0	2.7	.0	72.3	100	100	100	Do.
71	Do.	2-28	.6	1.7	14.3	12.1	2.9	8.6	.0	59.8	.....	Above	Above	Do.

APPENDIX A - Continued

Analyses of manhole atmospheres, Boston, 1934 - Continued

Manhole No.	Town	Date, 1934	Composition, percent by volume: CO_2	Illuminants	O_2	H_2	CO	CH_4 + C_2H_6	Gasoline + other hydrocarbons	N_2	Calculated explosibility	Explosibility by detector: Without filter	With filter	Weather
77	Boston	2-1	1.2	0.6	12.9	2.5	0.4	5.2	0.0	77.2	.....	Above	100	Cloudy.
77	Do.	6-6	1.8	.1	17.6	.0	.0	1.0	.0	79.5	20	25	25	Clear.
85	Do.	3-5	.8	.7	15.9	8.2	.8	6.2	.0	67.4	100	100	100	Do.
89	Do.	2-12	.3	.4	18.3	5.3	1.0	2.6	.0	72.1	100	100	100	Do.
90	Do.	2-12	.2	.3	19.0	3.6	.6	1.9	.0	74.4	100	100	100	Do.
91	Do.	2-12	.2	.3	19.3	2.3	.4	1.4	.0	76.1	75	75	75	Do.
92	Do.	2-12	.2	.3	19.1	3.4	.6	1.8	.0	74.6	100	100	100	Do.
94	Do.	1-25	.1	.1	20.4	.6	.1	.3	.0	78.4	20	25	25	Do.
95	Do.	3-22	.4	.5	18.2	3.9	.7	4.1	.0	72.2	100	100	100	Do.
95	Do.	3-26	.5	1.4	12.8	15.1	1.9	9.9	.0	58.4	.....	Above	Above	Do.
96	Do.	1-9	.1	.0	20.6	.0	.0	.0	.3	79.0	.....	25	00	Do.
97	Do.	1-9	.1	.0	20.6	.0	.0	.0	.2	79.1	.....	25	00	Do.
101	Do.	7-13	1.9	.4	15.6	.0	.0	4.3	.0	77.8	80	Above	25	Do.
105	Do.	3-1	.3	.6	17.8	6.4	.8	3.2	.0	70.9	100	100	100	Do.
106	Do.	3-1	.3	.6	18.1	6.0	.9	3.0	.0	71.1	100	100	100	Do.
107	Do.	3-1	.2	.4	19.8	2.3	.4	.8	.0	76.1	65	75	75	Do.
108	Do.	3-1	.2	.4	18.7	4.0	.6	1.6	.0	74.5	100	100	100	Do.
109	Do.	3-1	.4	.8	17.3	7.9	1.1	4.4	.0	68.1	.....	Above	100	Do.
110	Do.	3-1	.2	.3	19.4	3.4	.4	1.5	.0	74.8	95	100	100	Do.
111	Do.	3-1	.4	.9	17.6	6.7	1.1	4.4	.0	68.9	100	100	100	Do.
113	Do.	2-12	.3	.4	19.4	2.5	.4	1.4	.0	75.6	80	100	100	Do.
114	Do.	2-12	.4	.6	17.6	6.0	1.2	3.5	.0	70.7	100	100	100	Do.
116	Do.	3-1	.2	.5	18.9	4.2	.6	1.7	.0	73.9	100	100	100	Do.
117	Do.	3-1	.3	.4	18.6	4.9	.7	2.1	.0	73.0	100	100	100	Do.
119	Do.	7-10	2.0	.3	16.4	.0	.0	2.6	.0	78.7	50	50	50	Do.
122	Do.	1-10	.2	.4	18.8	5.3	.4	2.3	.0	72.6	100	100	100	Cloudy.
128	Do.	2-15	.2	.2	19.0	3.5	.5	2.1	.0	74.5	100	100	100	Clear.
129	Do.	2-15	.3	.2	18.8	3.4	.5	2.0	.0	74.8	100	100	100	Do.
130	Do.	2-15	.2	.4	19.4	2.6	.4	1.5	.0	75.5	80	100	100	Do.
131	Do.	2-15	1.0	1.4	13.8	12.8	1.7	8.8	.0	60.5	.....	Above	100	Do.
132	Do.	2-15	1.2	1.6	13.1	13.2	1.8	9.7	.0	59.4	.....	Above	Above	Do.
133	Do.	2-15	.8	.9	15.0	12.8	1.5	7.3	.0	61.7	.....	Above	100	Do.
134	Chelsea	2-13	.3	.3	18.4	5.0	.8	2.7	.0	72.5	100	100	100	Snow.
135	Do.	2-13	.1	.3	19.9	1.8	.4	.5	.0	77.0	50	50	50	Do.
136	Do.	5-7	1.3	.1	18.5	.6	.1	.6	.0	78.8	25	25	25	Clear.

APPENDIX A - Continued

Analyses of manhole atmospheres, Boston, 1934 - Continued

Manhole No.	Town	Date, 1934	Composition, percent by volume								Calculated explosibility	Explosibility by detector		Weather
			CO_2	Illuminants	O_2	H_2	CO	CH_4 + C_2H_6	Gasoline + other hydrocarbons	N_2		Without filter	With filter	
138	Dedham	8-9	0.4	0.3	20.0	0.4	Trace	0.9	0.0	78.0	25	25	25	Clear.
139	Framingham	10-15	.6	.4	19.2	.2	.1	1.2	.0	78.3	30	25	25	Rain.
140	Do.	8-15	.4	.3	20.1	.7	.1	.7	.0	77.7	30	25	25	Clear.
142	Newton	3-8	.5	.3	19.2	1.0	.2	.5	.0	78.3	35	25	25	Do.
145	Do.	3-8	.2	.4	19.1	2.9	.5	2.0	.0	74.9	95	100	100	Do.
146	Do.	3-8	.6	.5	18.6	1.8	.5	1.8	.0	76.2	75	100	75	Do.
148	Do.	3-8	.4	.6	17.9	6.4	.7	3.3	.0	70.7	100	100	100	Do.
150	Somerville	5-21	.4	.1	20.0	.7	Trace	.6	.0	78.2	25	25	25	Rain.
152	Stoneham	2-14	.3	.4	19.7	2.5	1.1	.8	.0	75.2	80	100	75	Clear.
153	Wakefield	2-14	1.5	2.2	14.2	13.1	5.8	5.1	.0	58.1	.....	Above	100	Do.
154	Do.	2-14	.9	1.3	16.4	10.0	3.7	3.2	.0	64.5	100	100	100	Do.
155	Do.	2-14	1.7	2.3	14.0	11.9	6.8	5.9	.0	57.4	.....	Above	100	Do.
156	Waltham	11-5	.4	.6	18.1	5.5	.5	3.3	.0	71.6	100	100	100	Do.
157	Wellesley	8-30	.6	.2	19.3	.6	.0	.9	.0	78.4	30	25	25	Do.
158	Winchester	3-12	.3	.1	19.7	.4	.1	.3	.0	79.1	15	25	25	Do.
159	Do.	3-12	.2	.2	19.8	.8	.2	.4	.0	78.4	25	25	25	Do.

Analyses of manhole atmospheres, Boston, 1934

(Representative manhole atmospheres of the New England Telephone & Telegraph Co.)

Manhole No.	Town	Date, 1934	Composition, percent by volume								Calculated explosibility	Explosibility by detector		Weather
			CO_2	Illuminants	O_2	H_2	CO	CH_4 + C_2H_6	Gasoline + other hydrocarbons	N_2		Without filter	With filter	
1	Boston	7-19	1.8	0.2	17.8	0.5	Trace	1.6	0.0	78.1	40	25	25	Clear.
2	Do.	7-9	4.0	.3	10.3	.9	Trace	4.1	.0	80.4	0.D.	0.D.	0.D.	Do.
2	Do.	10-9	2.2	.2	15.6	.7	Trace	2.7	.0	78.6	60	75	50	Do.
4	Do.	3-30	1.0	.8	15.0	7.0	0.6	4.8	.0	70.8	100	100	100	Do.
5	Do.	3-5	.9	.4	15.3	4.4	.5	4.2	.0	74.3	100	100	100	Do.
7	Do.	3-5	.7	.6	17.0	1.9	.3	2.1	.0	77.4	80	100	100	Do.
7	Do.	11-5	1.4	.1	16.9	.0	.0	1.8	.0	79.8	30	25	25	Do.
10	Do.	7-5	1.2	.6	16.2	.8	.1	6.3	.0	74.8	100	100	100	Do.
14	Do.	5-2	1.0	1.7	14.5	11.0	1.9	11.1	.0	58.8	.....	Above	Above	Do.
16	Do.	5-3	.6	.5	17.2	2.5	.1	1.5	.0	77.6	75	100	75	Cloudy.
17	Do.	5-3	.4	.4	19.3	.9	.0	1.2	.0	77.8	40	50	75	Rain.
18	Do.	3-7	1.0	.4	15.7	3.6	.4	4.2	.0	74.7	100	100	100	Clear.
20	Do.	3-7	1.0	.9	15.6	4.5	.6	4.4	.0	73.0	100	100	100	Do.
21	Do.	3-7	.2	.2	19.5	1.1	.2	.9	.0	77.9	40	50	50	Do.
27	Do.	2-15	.4	.2	19.2	1.5	.3	1.1	.0	77.3	50	50	50	Do.
28	Do.	2-12	.4	.4	18.6	2.6	.6	1.9	.0	75.5	90	100	75	Do.
31	Do.	12-24	.3	.2	19.6	2.1	.1	1.1	.0	76.6	60	75	50	Do.
32	Do.	4-11	.7	.5	16.3	.8	.1	3.3	.0	78.3	80	100	100	Do.
33	Do.	2-15	1.1	2.0	10.8	23.9	2.6	13.8	.0	45.8	.....	Above	Above	Do.
34	Do.	3-6	1.1	1.2	10.0	4.9	.4	8.5	.0	73.9	.....	Above	Above	Do.
34	Do.	7-11	6.4	.5	8.9	.4	Trace	11.2	.0	72.6	.....	Above	Above	Do.
35	Do.	3-6	.8	.9	13.1	8.8	.6	7.8	.0	68.0	.....	Above	50	Do.
36	Do.	6-14	1.8	.3	17.2	.1	.0	2.4	.0	78.2	45	50	50	Do.
36	Do.	10-2	1.2	.2	18.6	.0	.0	1.3	.0	78.7	25	25	25	Do.
37	Do.	3-6	1.7	.4	15.4	.4	.1	1.2	.0	80.8	35	25	25	Do.
40	Do.	3-6	1.1	.4	15.2	.7	.1	2.6	.0	79.9	65	75	50	Do.
41	Do.	3-6	1.3	.4	15.6	.5	.1	1.9	.0	80.2	50	50	25	Do.
42	Do.	3-6	.7	.3	16.1	1.1	.2	3.2	.0	78.4	80	100	75	Do.
45	Do.	7-11	2.9	.1	15.8	.0	.0	1.1	.0	80.1	20	25	25	Do.
48	Do.	11-1	1.5	.3	16.1	2.6	.2	2.3	.0	77.0	90	100	100	Rain.
49	Do.	7-11	1.9	.2	17.4	.4	Trace	1.8	.0	78.3	40	50	25	Clear.
50	Do.	10-9	1.4	.2	17.3	2.6	.3	2.2	.0	76.0	90	100	100	Do.
51	Do.	11-1	1.2	.2	17.3	1.6	.1	1.6	.0	78.0	60	75	50	Rain.
52	Do.	11-1	1.2	.3	17.0	3.1	.2	2.6	.0	75.6	100	100	100	Do.
53	Do.	11-1	.9	.4	17.0	4.3	.4	3.3	.0	73.7	100	100	100	Do.
54	Do.	11-1	1.3	.3	16.7	2.0	.2	2.0	.0	77.5	75	100	50	Do.
55	Do.	11-1	1.4	.3	16.7	1.8	.1	2.0	.0	77.7	70	100	50	Do.
56	Do.	3-15	.9	.2	17.3	.9	.1	1.7	.0	78.9	50	50	50	Clear.
63	Do.	7-24	8.7	.4	7.6	.0	.0	6.1	.0	77.2	0.D.	0.D.	0.D.	Do.

Analyses of manhole atmospheres, Boston, 1934 - Continued

Manhole No.	Town	Date, 1934	Composition, percent by volume: CO_2	Illuminants	O_2	H_2	CO	CH_4 + C_2H_6	Gasoline + other hydrocarbons	N_2	Calculated explosibility	Explosibility by detector: Without filter	With filter	Weather
63	Boston	11-6	1.8	0.3	16.0	0.0	0.0	3.2	0.0	78.7	60	75	25	Rain.
64	Do.	7-23	3.2	.2	16.0	.0	.0	1.8	.0	78.8	35	50	50	Clear.
68	Do.	10-8	.5	.1	19.8	.8	.1	0.6	.0	78.1	25	25	25	Do.
68	Do.	11-6	.8	.3	17.8	3.2	.2	2.6	.0	75.1	100	100	100	Rain.
69	Do.	11-6	1.4	.3	16.3	1.0	.1	2.9	.0	78.0	70	100	25	Cloudy.
72	Do.	11-16	1.3	.6	14.8	9.8	.8	6.2	.0	66.5	.....	Above	100	Clear.
73	Do.	7-6	2.1	.5	14.5	1.7	.1	4.9	.0	76.2	100	Above	100	Do.
76	Do.	7-6	.3	.1	20.2	.7	.1	.5	.0	78.1	25	25	25	Do.
77	Do.	6-13	4.3	.3	13.8	.0	.0	1.1	.0	80.5	0.D.	25	25	Cloudy.
80	Do.	2-15	.3	.3	18.1	4.4	1.0	2.1	.0	73.8	100	75	50	Do.
81	Do.	2-15	.3	.3	19.3	2.2	.3	1.1	.0	76.5	65	75	75	Do.
82	Do.	2-15	.2	.3	19.6	2.6	.5	1.2	.0	75.6	75	75	75	Do.
83	Do.	6-6	6.9	.2	9.4	.0	.0	1.8	.0	81.7	0.D.	0.D.	0.D.	Clear.
84	Do.	4-3	1.6	.2	16.4	.5	Trace	.8	.0	80.5	25	25	25	Do.
84	Do.	6-6	1.9	.7	16.6	3.2	.2	2.2	.0	75.2	100	100	100	Do.
85	Do.	3-29	.6	.3	17.9	1.7	.1	1.8	.0	77.6	65	50	50	Do.
88	Do.	3-29	.4	.2	18.7	1.8	.2	1.4	.0	77.3	60	50	50	Do.
90	Do.	3-13	.3	.2	19.5	1.8	.3	1.2	.0	76.7	60	50	50	Do.
93	Do.	3-27	.9	.2	16.6	.7	.1	1.5	.0	80.0	40	50	50	Do.
93	Do.	6-5	6.8	.9	4.6	.0	.0	6.2	.0	81.5	0.D.	0.D.	0.D.	Do.
94	Do.	6-5	6.6	.6	5.3	.0	.0	8.5	.0	79.0	0.D.	0.D.	0.D.	Do.
94	Do.	7-9	9.0	.4	3.5	.0	.0	12.1	.0	75.0	.....	0.D.	Above	Do.
95	Do.	3-13	.5	.3	17.8	1.6	.2	1.9	.0	77.7	65	75	75	Do.
96	Do.	3-29	.8	.4	17.5	2.2	.3	2.5	.0	76.3	90	100	100	Do.
96	Do.	7-10	2.6	.1	16.5	.0	.0	.4	.0	80.4	10	25	25	Do.
98	Do.	6-15	2.5	.4	15.5	.4	Trace	1.4	.0	79.8	35	25	25	Do.
100	Do.	1-9	1.3	.1	16.3	.4	.1	.8	.0	81.0	25	25	25	Do.
102	Do.	1-12	.9	.5	14.9	5.4	.5	5.1	.0	72.7	100	100	100	Do.
102	Do.	6-8	3.0	.5	13.4	1.8	.3	2.5	.0	78.5	0.D.	0.D.	0.D.	Do.
105	Do.	5-25	.6	.3	18.9	.9	.1	1.4	.0	77.8	45	50	50	Do.
109	Do.	6-1	.6	.4	17.9	4.8	.4	3.2	.0	72.7	100	100	100	Do.
111	Do.	6-1	.7	.5	18.0	4.4	.4	3.1	.0	72.9	100	100	100	Do.
113	Do.	4-3	2.8	1.1	9.5	2.0	.2	6.1	.0	78.3	.....	Above	Above	Do.
114	Do.	6-6	5.0	1.2	8.1	.1	.0	14.3	.0	71.3	.....	0.D.	0.D.	Do.
120	Do.	9-25	2.0	.4	15.8	.3	.0	4.7	.0	76.8	0.D.	0.D.	0.D.	Do.
124	Do.	10-8	3.6	.3	13.2	.7	.0	2.8	.0	79.4	0.D.	0.D.	0.D.	Do.
124	Do.	11-6	1.4	.3	16.7	1.3	.1	2.6	.0	77.6	70	75	75	Rain.
126	Do.	7-3	3.1	.1	16.0	.0	.0	.9	.0	79.9	15	25	25	Clear.
129	Do.	6-15	1.9	.2	15.3	.9	.1	2.2	.0	79.4	55	50	25	Do.
129	Do.	7-6	1.5	.2	17.0	.5	.0	2.1	.0	78.7	45	50	50	Do.
130	Do.	4-2	.9	1.5	13.6	14.5	1.8	12.4	.0	55.3	Above	Above	Above	Do.
132	Do.	11-5	.8	.4	17.5	2.8	.3	3.9	.0	74.3	100	100	50	Do.

APPENDIX B - Continued

Analyses of manhole atmospheres, Boston, 1934 - Continued

Manhole No.	Town	Date, 1934	Composition, percent by volume								Calculated explosibility	Explosibility by detector		Weather
			CO_2	Illuminants	O_2	H_2	CO	CH_4 + C_2H_6	Gasoline + other hydrocarbons	N_2		Without filter	With filter	
133	Boston	6-21	3.1	0.9	13.1	3.1	0.3	5.9	0.0	73.6	.....	Above	100	Clear.
134	Do.	6-21	3.1	.8	12.6	4.2	.4	7.1	.0	71.8	.....	Above	Above	Do.
135	Do.	4-17	.2	.1	19.7	.5	Trace	.9	.0	78.6	25	25	25	Do.
138	Do.	7-10	5.0	.3	5.9	.0	.0	15.2	.0	73.6	.....	Above	Above	Do.
139	Do.	7-10	2.7	.3	14.4	.0	.0	2.9	.0	79.7	55	50	0.D.	Do.
143	Do.	6-6	6.2	.5	8.1	.0	.0	3.5	.0	81.7	0.D.	0.D.	0.D.	Do.
146	Do.	6-21	3.7	.5	12.7	2.5	.3	5.8	.0	74.5	.....	Above	Above	Do.
148	Do.	6-13	1.7	.3	17.0	1.1	.2	1.3	.0	78.4	50	50	50	Cloudy.
151	Do.	1-15	.7	.2	17.5	1.1	.2	1.1	.0	79.2	45	50	50	Clear.
151	Do.	6-13	3.4	.3	14.5	.0	.0	1.4	.0	80.4	0.D.	0.D.	0.D.	Cloudy.
153	Do.	3-1	.1	.0	20.3	.7	Trace	.4	.0	78.5	20	25	25	Clear.
158	Do.	5-25	2.9	.3	15.4	.1	.0	2.3	.0	79.0	0.D.	0.D.	0.D.	Do.
161	Do.	6-7	6.5	.6	8.3	.0	.0	3.9	.0	80.7	0.D.	0.D.	0.D.	Cloudy.
163	Do.	7-11	3.7	.4	13.7	.4	.0	2.2	.0	79.6	0.D.	0.D.	0.D.	Clear.
165	Do.	6-6	1.3	.8	15.5	1.6	.1	3.6	.0	77.1	100	100	100	Do.
169	Do.	7-11	3.4	.2	15.5	.0	.0	.8	.0	80.1	15	25	25	Do.
171	Do.	6-6	.5	.2	19.9	.9	.1	1.0	.0	77.4	35	25	25	Do.
172	Do.	7-12	6.8	.3	8.4	.6	.1	3.1	.0	80.7	0.D.	0.D.	0.D.	Do.
174	Do.	4-11	1.9	.2	15.2	.2	.0	1.2	.0	81.3	25	25	25	Do.
179	Brookline	1-8	.5	.1	18.1	.4	Trace	.9	.0	80.0	25	25	25	Cloudy.
182	Do.	1-8	.4	.1	19.0	2.2	.3	1.5	.0	76.5	70	75	75	Do.
186	Do.	10-17	.8	.4	18.2	2.6	.2	2.2	.0	75.6	90	100	100	Clear.
187	Do.	10-17	1.6	.4	17.1	2.6	.2	2.3	.0	75.8	90	100	100	Do.
191	Do.	1-4	.1	.1	19.9	1.1	.2	.4	.0	78.2	30	50	25	Snow.
192	Do.	1-4	.1	.1	19.8	1.0	.1	.6	.0	78.3	30	25	25	Do.
193	Do.	1-4	.1	.1	20.1	.7	.1	.4	.0	78.5	20	25	25	Do.
194	Chelsea	2-13	.2	.2	20.3	1.0	.2	.3	.0	77.8	30	25	25	Cloudy.
196	Lexington	9-13	1.1	.1	19.0	.2	.0	.6	.0	79.0	15	25	25	Clear.
197	Natick	10-15	.5	.1	19.9	.2	Trace	.6	.0	78.7	15	25	25	Rain.
198	Newton	3-8	.2	.1	20.1	.8	.2	.5	.0	78.1	25	25	25	Clear.
201	Somerville	5-29	3.6	.4	11.1	1.9	.2	6.5	.0	76.3	.....	Above	Above	Do.
203	Do.	5-28	1.6	.8	15.5	7.4	.9	4.9	.0	68.9	.....	Above	100	Cloudy.
204	Wakefield	2-14	.9	.4	18.9	1.7	.9	.7	.0	76.5	60	75	50	Clear.
205	Watertown	3-8	.2	.2	20.0	.8	.1	.5	.0	78.2	25	25	25	Do.
206	Do.	3-8	.5	.3	18.1	3.2	.4	2.8	.0	74.7	100	100	100	Do.
207	Do.	9-11	1.2	.2	18.8	.7	Trace	.7	.0	78.4	25	25	25	Do.
209	Wellesley	8-30	1.4	.1	19.1	.2	.0	.5	.0	78.7	15	25	25	Do.
210	Winchester	9-24	.5	.1	19.9	.1	.0	.4	.0	79.0	10	25	25	Cloudy.
212	Woburn	9-21	1.4	.2	17.1	3.0	.2	2.6	.0	75.5	100	100	100	Clear.

After this report has served your purpose and if you have no further need for it, please return it to the Bureau of Mines. The use of this mailing label to do so will be official business and no postage stamps will be required.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

62.09
n32mo
Cop2

R. I. 3306 June 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

PROGRESS REPORTS - METALLURGICAL DIVISION

12. ANNUAL REPORT OF THE METALLURGICAL DIVISION, FISCAL YEAR 1935

BY

R. S. DEAN

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3306,
June 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PROGRESS REPORTS - METALLURGICAL DIVISION

12 Annual Report of the Metallurgical Division, Fiscal Year 1935[1]

By R. S. Dean[2]

INTRODUCTION

The function of the Metallurgical Division is to conduct investigations relating to the treatment of mineral products from the state in which they are mined to the refined metal or other unfabricated product. These investigations include scientific and practical studies of ore dressing, covering the crushing and grinding of ores, separation by gravity, flotation, and magnetic methods; the hydrometallurgical treatment of ores, including cyanidation and amalgamation of the precious metals; the smelting and refining of copper, lead, and zinc, as well as the rare metals; and steel-melting practices from the raw ore to the finished steel. The object of this work is to conserve natural resources and eliminate waste by furnishing information on current methods of metallurgical practice, making fundamental studies of metallurgical processes, and developing new processes to the point of demonstrating their practicability. The Division is made up of sections devoted to the several phases of this work, each section being headed by an investigator capable of supervising his particular work. The work is carried on at the experiment stations and field offices or in the field, whichever seems best suited for the work in hand.

The following projects should be mentioned as particularly outstanding among those consummated successfully during the year:

1. Application of the type of chemicals known in the textile industry as wetting-out agents to flotation and agglomeration of ores, particulary soluble salts such as sodium and potassium chloride which must be treated in saturated brine.

2. Demonstration of a linear relation between surface and coercive force in magnetic minerals and its application to magnetic separation and to the determination of grinding efficiencies and other metallurgical data which require surface determination at very fine sizes.

3. Pilot-plant demonstration of the practicability of the reduction of iron ore by natural gas and conversion of this direct iron into steel by direct melting and by forging and rolling before melting. The structure and properties of the iron so produced were studied in detail.

This report covers the fiscal year July 1, 1934, to June 30, 1935.

ACKNOWLEDGMENT

This report has been prepared from material submitted by the following members of the Metallurgical Division staff:

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3306."

2 Chief engineer, Metallurgical Division, U.S. Bureau of Mines.

C. W. Davis, Assistant to the Chief Engineer.
C. G. Maier, Supervising Engineer, Metallurgical Fundamentals Section.
V. H. Gottschalk, Supervising Engineer, Special Studies Section, Supervising Engineer, Copper Metallurgy Section.
W. H. Coghill, Supervising Engineer, Ore Dressing Section.
G. L. Oldright, Supervising Engineer, Precious Metals Section, and Supervising Engineer, Lead and Zinc Section.
T. L. Joseph, Supervising Engineer, Iron and Steel Section.

Acknowledgment is made of assistance rendered by the following cooperating institutions:

Missouri School of Mines and Metallurgy.
University of Minnesota.
University of Utah.
University of Nevada.
University of California.
University of Arizona.
University of Alabama.
University of Maryland.

METALLURGICAL FUNDAMENTALS

The work of the metallurgical fundamentals section is about equally divided among three distinct but related and overlapping types of investigations.

1. The section seeks to supply needed thermodynamic constants for important metallurgical materials by means of direct physicochemical measurements. The foremost example of this type of work is its "continuous program" investigation of low-temperature specific heats. From such data entropy values for the material in question are derived; these in turn permit estimation of the course of metallurgical or chemical reactions and are now often used to study reactions where direct experimentation is difficult, costly, or impossible. The Division truly has been a pioneer in the application of these methods to metallurgical problems; that their efficacy is becoming recognized is attested by the fact that several universities and technical schools recently have undertaken to install equipment for similar measurements. Until the last few years the laboratory at Berkeley was the only one in the country equipped for this work, which had the direct practical application in view, although a few other laboratories were busy with academic investigations in the same field.

Another example of a similar type of work, but delegated to interest in a particular system, is the study now in progress of the physical-chemical properties of chromium chlorides. These investigations were undertaken because preliminary experiments indicated clearly that the beneficiation of domestic chromite ores may be made feasible and that the production of low-carbon chromium may be facilitated by processes involving chromic chlorides and because existing data on the chemistry of these chlorides was meager and largely incorrect.

A final example of direct experimental measurement is the precise determination of the changes of density and coercive force produced by the action of cold work on copper and iron, undertaken to throw light upon some of the fundamental physical problems involved in the crushing of ores.

2. The section also undertakes correlative studies of data on thermodynamic and physical chemical properties of metallurgically important materials. Operating metallurgists and even technical research men rarely have the facilities or opportunity to correlate such

figures. It is the purpose of this work to supply figures, as authentic as is possible, for the properties that are technically important, in a form that may be readily understood and utilized with more confidence than any individual selection uncritically selected from the literature. Only in recent years have metallurgists begun to use chemical-thermodynamic methods, and while such methods have proved thoroughly practical and useful their widespread application has been somewhat hampered by lack of ready figures and facility in their use.

An example of such activity is the preparation of a bulletin dealing with the heats of fusion of inorganic materials, as derived from melting-point data. Similar reports submitted for publication during the year deal with the properties of metal carbonates and the vapor pressures of inorganic materials.

The work on the chemistry of chromium chlorides, mentioned in the preceding section, is being amplified to include data on other compounds of this metal.

3. The section attempts to carry further the results of work described in sections 1 and 2 to the initial stages of applying them to the improvement of existing processes or to the development of new processes. Although limitations of personnel and equipment prevent any ambitious attempt to apply the scientific and technical results on semicommercial or pilot-plant scale, nevertheless there is a gap between the thermodynamic and physical-chemical data in themselves and in their actual application to industrial processes. The work required to obtain and correlate these figures is justified only if their application to the problems of the industry can be shown clearly.

The best example of such continuation is the application of thermal and thermodynamic figures to the problem of producing sponge iron from pyrite cinder and natural gas. Having carried the work in previous years to a point requiring industrial participation, the section has cooperated informally during the last year with a particular member of the industry especially interested in continuing the investigations through the small pilot-plant stage.

In other work, such as that on natural carbonates described in the preceding section, it was not practical to apply the results of theoretical calculations, but the data obtained were utilized to show the unsoundness of a number of proposed schemes for the beneficiation of rhodochrosite ores and, further, to disclose the essential features of a new method, as yet untried, which offered the most promise chemically and economically.

Low-Temperature Specific-Heat Investigation

Because the metallurgical fundamentals laboratory of the Bureau is recognized as the foremost one making low-temperature investigations of special interest to metallurgists, it receives rather frequent requests or suggestions from the industry for measurements on specific materials. If such suggestions are timely or refer to materials of rather general importance an effort is made to complete the required measurements as promptly as possible.

During the year members of the research department of the United States Steel Corporation suggested the advisability of measurements on the cristobalite and tridymite modifications of quartz. Certain problems relating to siliceous refractories involve problems pertaining to the transformation between the crystal varieties of silica; such information is rather difficult to obtain by direct experiment because of the relative slowness with which the changes may take place. Measurements were therefore completed upon those materials.

The reducibility relationships between the oxides of vanadium and the metal are another example of the reliability of thermodynamic methods over direct experimentation. Although measurements upon the oxides of vanadium had already been completed, during the last year a sample of pure vanadium metal was made available through the Vanadium Corporation of America. The fact that a pound sample was valued at $500 shows clearly that despite the wide use of this metal in alloys much may still be accomplished in making pure metal commercially usable.

Measurements were completed upon the metal, and the results were correlated with the previous measurements on oxides and with direct-reduction experiments described in technical literature. It was shown that the direct measurements were at least a thousandfold in error and that precise direct tests would have required an unattainable accuracy of chemical analysis. The figures not only show that vanadium is more difficult to reduce than had been indicated by high-temperature direct experiments but also furnish quantitative data from which the degree of reducibility at any temperature may be correctly estimated and thus serve as the initial step in suggesting or rejecting proposed methods of reduction.

During the year measurements were begun upon a series of various hydrates and polymorphic forms of calcium sulphate. Here, again, the suggestion has come from the industry, in this instance with particular reference to the need for fundamental data to study critically the calcination of plaster. Low-temperature measurements will furnish "key" information relative to the whole series of properties involved in various stages of calcination and will further furnish thermal data for setting up thermal-efficiency balances of the manufacturing process.

Of more theoretical interest, but also with commercial implications, are the measurements being made upon selenium and tellurium. Theoretically they are important because these elements occur as both crystalline and vitreous solids - a condition much rarer for elements than for compounds - and because, when combined as selenides and tellurides of manganese, anomalous behavior of specific heat at low temperatures is expected. Such anomalies, not as yet explained or utilized, are common in materials having magnetic properties. Practically, selenium is a valuable decolorizer for glass; the low condition of the copper industry, whose refining process furnishes most of the available supply, combined with rather good capacity in the glass industry, has resulted in a modest shortage of this element, in spite of the fact that only minute amounts per ton of glass are required.

Thermodynamics of Chromium Reduction

Low-temperature specific-heat measurements completed last year upon chromium and its trioxide show conclusively that direct methods for the production of low-carbon chromium from oxide ores, using gaseous or solid reductants, are limited by chemical-equilibrium conditions to temperature ranges above 1,500°C. and conditions where only electric furnaces are suitable. For such purpose only the best chromite ores are suitable, and domestic grades, while plentiful, cannot readily compete with imported material.

When, as explained in an earlier part of this report, preliminary experiments showed that the chromium could be volatilized from chromite ores at temperatures near 800°C., in the form of chlorides, and that these chlorides might be reduced at similar temperatures to pure chromium it became evident that a thorough study of vapor and dissociation pressures, equilibria of reduction, and other thermodynamic relationships was entirely justified by the potentialities of the chlorine metallurgy of chromium.

During the year extensive measurements were completed upon such properties. The chemistry of at least three chlorides of chromium is involved in such a problem. In addition to the known forms of dichloride and trichloride, experiments proved the existence in the gaseous phase of a hitherto unknown tetrachloride. Each of these is interrelated with the other forms by a series of dissociation and vapor-pressure relationships, and the disentangling of the observed behavior into a coordinated series of thermodynamic relationships is an extremely complex problem. This work has been nearly completed by measurements on the vapor pressures of chromous and chromic chloride, the dissociation pressures of chromium tetrachloride, and the hydrogen-reduction equilibria of chromous chloride. The remaining item required to tie together the properties of all these materials is the dissociation pressure of chromic chloride; experiments to elucidate this behavior are in progress.

The direct practical problem of applying the theoretical information obtained to the actual chlorination process was undertaken late in the year, after enough theoretical information had been collected to apply the tests intelligently. Results so far have been extremely encouraging from several points of view and have disclosed only difficulties of a mechanical rather than chemical nature. Thus, the speed of the chlorination near 800°C. is entirely adequate, the recovery of chromium high, and the reaction has been found to be exothermic to a degree warranting the conclusion that on a large scale it may be carried on spontaneously after once started, without the necessity of transmitted heat. Because of the nature of the proposed chlorination this is a very important item. The difficulties disclosed are largely those of handling and recovering the bulky sublimate of chromic chloride.

Densities of Cold-Worked Metals

The question of the minimum amount of work energy required to comminute a solid (equivalent to thermodynamic crushing efficiency) cannot be answered until definite information is available as to the amount of such energy stored in latent form in the comminuted material. Earlier experiments showed that latent energy could be proved for metals but that it could not be proved for a typical ionic-type crystal lattice (calcite) because the crushing resulted in changes of chemical composition which obscured the direct determination of the stored energy.

This problem is of such major importance, both from the theoretical viewpoint of knowledge of crystalline materials as well as practically in connection with crushing problems, that further investigation was warranted. It can be shown thermodynamically that changes in the lattice parameter, detectable as minute changes in density, should be concomitant with the storage of energy in comminuted materials, and for such studies metals are better suited than rocklike materials. During the year an extensive series of precise density determinations were completed upon samples of copper and iron which had been subjected to a variety of degrees and methods of cold working; in the case of the latter material magnetic coercive forces were also determined.

These measurements showed that the changes of density were much more complex than had been supposed from the scattered measurements in technical literature. It can be shown that the observed density changes of copper are consistent with lattice distortion observed in cold working and as determined by X-ray methods. The change of density of iron also was shown to be associated with the formation of new internal crystal surfaces by demonstration that under certain conditions the density and coercive force showed linear relationships with increasing degrees of cold work.

If both of these conclusions are sound, as indicated by the experimental data obtained, an entirely new viewpoint concerning the nature of changes in cold-working metals is a logical necessity, which, briefly, may be stated to rest on the hypothesis that lattice distortion is the result rather than the source of the changes produced physically by cold working. aith such a starting point it is possible to develop a theory that explains successfully a number of observed phenomena previously interpreted by divergent and contradictory theories. The experiments, which have progressed as far as justifiable for Bureau purposes and which are now being prepared for publication, may be described as having elicited information of fundamental importance to the theory of the metallic and crystalline state.

Heats of Fusion

The direct experimental determination of heats of fusion of metals and inorganic materials is extremely difficult when the melting points are relatively high and few direct

data are available. The Metallurgical Division has received a large number of requests for information regarding heats of fusion of certain important metallurgical materials required to set up heat balances for standard metallurgical smelting processes. The recalculation of melting-point data for binary systems offers a means of calculating heats of fusion for those materials that make nearly perfect solutions with each other in the liquid state. The results of a few calculations of this type are given in standard handbooks, but no systematic attempt heretofore has been made to examine the great mass of available experimental data critically and to correlate all of the results with a view of obtaining heats of fusion. It has usually been assumed that most of these systems produce very imperfect solutions. In the course of the year, after an extensive study of the literature, which required reference to some 2,500 scientific articles, it was found that a much higher percentage of materials than had been supposed gave melting-point data sufficiently close to those of a perfect solution to justify the calculation of heats of fusion. A complete series of these calculations has been carried out, the results of which are nearly completed and soon will be described in a publication. The calculations enable heats of fusion to be given for a great many materials in which direct determinations are not available, they confirm a number of values that were previously in doubt, and, finally, they permit the classification of melting-point systems into groups representing nearly perfect solutions and those in which the calculations show that the data could not have been complete and that probably solid solutions and other unexpected complications exist.

When the report of these calculations is published the metallurgist will have readily available and digested data embodying all previously known heats of fusion; these data, in conjunction with Contributions to Theoretical Metallurgy, which present specific-heat values in a similar manner, will greatly facilitate the determination of the thermal balance and efficiency of the processes of interest to practicing process metallurgists. Requests for the use of the data in advance of actual publication have already been received in connection with study of the copper-smelting process from a thermal viewpoint.

Since the matter of thermal efficiency of high-temperature metallurgical processes is very important economically and since nearly all such processes are subject to improvement in this particular respect it is evident that the presentation of all available information on the subject, condensed to the most authentic and readily usable form, is of immediate utility and importance.

Metal Carbonates

Some years ago the attention of the Division was directed by members of the War Department to the problem of the beneficiation of domestic rhodochrosite ores (manganese carbonates), large tonnages of which are available in the Northwest but are low-grade. The carbonates of calcium and magnesium are intimately associated with those of manganese in these ores. In point of gross tonnage they may constitute our largest single supply of manganese, but all attempts at their beneficiation have been unsuccessful.

In studying this problem the section has adopted the rational and scientific method of first obtaining fundamental information concerning the thermodynamic properties of important metal carbonates and then applying the information to investigate the chemical soundness of methods that have been attempted in the past; as is not unusual, the calculations made on this basis have revealed a new but untried process which seems to be theoretically the best that has been proposed.

Thus, in previous years an extensive series of low-temperature specific-heat measurements of the common metallic carbonates, which are present in typical carbonate ores, was completed; during the present year the entropy values so derived were applied to existing

data on the dissociation and solubility of carbonates. After the information on the behavior of carbonates had thus been authenticated and correlated it was easy to show that most of the processes that have been tried on rhodochrosite and that had sufficiently low material and energy costs were chemically unsound in principle.

Further, it was shown to be theoretically possible to replace the manganese in the ore by calcium; this suggests a leaching scheme using a soluble calcium salt (such as chloride), dissolving the manganese, leaving lime, and later recovering the chloride ion by again precipitating the manganese by lime, which may be simply a portion of calcined ore. Such a scheme obviously implies chemical control, and the fundamental data were applied to show the limitations and possibilities inherent in the process. The method would be self-containing in materials, except for inevitable loss of some chloride ion in wash waters. For this reason it seems to be more feasible economically than other processes requiring the use of external leaching or solution agents. A complete report including not only fundamental data but also details of application has been prepared for publication as part of the series Contributions to Theoretical Metallurgy.

Direct Production of Steel

The experiments on sponge iron, while part of a larger general program, were of specific and local interest. Some years ago the availability of cheap natural gas and local accumulations of more than 3/4 million tons of pyrite cinder with upward of 400,000 tons of metallic iron content made a renewed attempt to treat such material a promising problem. Previous attempts to utilize Bureau of Mines work, using solid carbonaceous materials as reductants, had not proved feasible economically because of local conditions with respect to fuel. Founded on thermodynamic calculations and a later direct experimental check on a basis of 200 pounds per day, a process was proposed which consisted of (1) catalytically re-forming natural gas-preheated air mixtures to an active reducing gas containing carbon monoxide and hydrogen and (2) then utilizing the hot re-formed gas for reduction of the cinders.

During the year the Mountain Copper Co., at its Martinez plant, has operated a unit with a capacity of 1 ton per day which utilizes this process, and the section has informally cooperated in this project. In return for the Bureau's help the operating reports of the plant have been turned over to the section, and a complete publication including them is being prepared.

The accomplishments may be summarized briefly as follows:

The metallurgical soundness and practicability of the method have been demonstrated.

Figures have been obtained that will serve as a fairly adequate guide in estimating costs in a production plant.

The work has culminated in the melting in a commercial open-hearth furnace of a lot of 20 odd tons of sponge iron, with the production of a grade of steel slightly superior to the average produced by the unit when fed with scrap and pig iron.

The cost figures are particularly interesting. If natural gas is assumed to cost $0.15 per thousand cubic feet direct extrapolation of data from the plant producing 1 ton per day indicated that the costs for a plant producing 20 tons should range from $16 to $18 per ton of metallic iron produced. On a basis of 100 tons per day, the figure drops to $12 to $14 per ton. Such costs are comparable with local pig-iron and scrap-iron prices at the present time, and any improvement tending to bring such prices to a more nearly normal level would justify the immediate production of sponge iron as a substitute for these materials under local conditions.

The melting test, which was carried out under strictly commercial operating conditions, showed that most of the expected difficulties in melting and refining were not serious with

the well-reduced material that can be supplied by this process; in fact, certain economies in melting might be expected from the use of sponge iron.

An extremely high-grade sponge iron cannot be produced from so impure a source of material as pyrite cinder; whence the outlet for this grade is limited to competition with the lowest-priced forms of commercial iron, such as scrap and pig. It is possible, however, that sound engineering design might so lower the costs given above (the actual fuel cost was shown to be less than $5 per ton) that it would even now be possible to produce sponge to substitute economically for scrap.

The following contributions report work completed by the metallurgical fundamentals section during 1933-34 but not published in time to be included in the Director's Annual Report:

Doerner, H. A., Oldright, G. L., Brighton, T. B., and Dice, C. L. Progress Reports - Metallurgical Division. 8. Studies in Zinc. Rept. of Investigations 3256, Bureau of Mines, December 1934, pp. 1-43.

Maier, C. G. Progress Reports - Metallurgical Division. 9. Thermodynamic Data on Some Metallurgically Important Compounds of Lead and the Antimony-Group Metals and Their Applications. Rept. of Investigations 3262, Bureau of Mines, December 1934, pp. 1-54.

Maier, C. G., and Anderson, C. T. Disposition of Work Energy Applied to Crystals. Jour. Chem. Phys., vol. 2, no. 8, August 1934, pp. 513-527.

Anderson, C. T. The Heat Capacities at Low Temperatures of the Oxides of Strontium and Barium. Jour. Am. Chem. Soc., vol. 57, March 1935, pp. 429-431.

SPECIAL STUDIES SECTION

The special studies section of the Metallurgical Division seeks to advance metallurgical art by discovering and developing radically new processes in milling, smelting, and refining.

Its active problems for the fiscal year 1934-35 were as follows:

1. Explosive shattering.
2. Magnetic properties of minerals.
3. Electrical properties of minerals.
4. Gas absorption on minerals.
5. Dust settling.

1. Explosive Shattering

Explosive shattering consists essentially in steaming a confined charge for a short period and suddenly releasing the pressure, thereby causing the material to be ejected explosively with a shattering effect.

Previous reports have shown the effect of such variables as steam pressure, size of charge, and length of steaming period.

That the impact of the ejected charge against a solid obstruction increases shattering has been noted, but until recently no comparative figures were available. The effect of the distance at which impact took place was determined on plus 48-mesh dolomite for three different impact distances. The results which follow show that shattering is increased about 50 percent when the impact distance is decreased from 9 feet to 1 foot:

Impact distance, feet	Percent of size produced					
	150 pounds pressure			250 pounds pressure		
	48/100 mesh	100/200 mesh	-200 mesh	48/100 mesh	100/200 mesh	-200 mesh
9	8.8	5.5	3.6	12.6	8.1	5.3
3 1/4	11.6	7.3	4.8	15.9	10.2	6.6
1	12.2	7.9	5.3	18.0	12.1	8.0

The effect of impact distance was exemplified further in the shattering of graphite, the object being to produce as much minus 400-mesh size as possible. The results which follow include those obtained with a special small-discharge machine impacting at 3-1/2 inches.

Impact distance	Percent of minus 400-mesh produced
3 feet 3 inches	10.4
1 foot	14.2
3-1/2 inches	19.7

The shattering due to impact has been a factor in the recent development of the explosive machine.

Many examples cited on the explosive results of different ores show in general that a more preferential crushing is obtained with less fine material when the ore is explosively shattered. While very little work has been done during the past year on ores the results obtained on a pyritic gold ore are of interest. This ore, which was essentially an arsenical pyrite, was exploded after igniting to incipient arsenic volatilization. The results given here show the increased ease of shattering as well as the greatly improved extraction by cyanidation.

Treatment	-200 mesh produced, percent	Gold recovery by cyanidation on -200 mesh
Ordinary ball-mill grind	25.5	63.8
Explosive shattering, on raw ore	24.7	65.5
Explosive shattering, on ignited ore	32.6	81.1

A special explosive machine with a small discharge at the top was used to obtain data for comparison with the full-opening discharge of the regular machine. The following results were obtained on 400-gram charges:

	Percent of size produced			Percent in -48 mesh		
	48/100	100/200	-200	48/100	100/200	-200
Regular machine:						
2-3/16-inch discharge	22.4	16.8	37.6	29.2	21.9	48.9
3/4-inch discharge	21.2	14.4	31.2	31.7	21.6	46.7
1/2-inch discharge	22.8	16.0	34.0	31.3	22.0	46.7

The table indicates that small discharges have nearly as much shattering effect as larger openings and that the product is somewhat more granular.

The machine with the small opening was not adapted to an extended investigation; the latest design has therefore been equipped for variable discharges. The mechanical advantage of smaller discharges in a lighter mechanism and a tighter steam seal is important enough to warrant further study of this phase.

The effect of explosively ejecting material from a nozzle, either operated continuously or intermittently, gave some interesting results which are compared here with best-capacity and best-efficiency results obtained with the regular explosion machine:

	Impact distance, inches	Steam consumption, pounds per ton of charge	-48 mesh product, percent	Steam cost per ton -48 mesh, cents	Percent in -48 mesh		
					48/100	100/200	-200
Regular explosive machine:							
Best capacity	6	277.5	19.20	18.44	29.2	21.9	48.9
Best efficiency	6	53.5	13.60	5.00	31.6	22.1	46.3
Nozzle, intermittent:							
Best capacity	5	725	20.22	45.70	21.1	14.0	64.9
Best efficiency	5	355	11.20	40.46	31.3	20.5	48.2
Nozzle, continuous:							
Best capacity	5	1,135	19.58	73.90	23.9	17.4	58.7
Best efficiency	5	375	14.13	33.83	28.5	20.1	51.4

Were it not for the high steam consumption the nozzle machine would be an exceedingly simple apparatus for shattering. In the tests giving best results for efficiency the capacity of the nozzle, which can be held easily in one hand, was 3,000 pounds of charge or 420 pounds of minus 48-mesh per hour. The high steam consumption, however, is not encouraging.

The possibilities of economic explosive shattering of ore with very low steam consumption, as shown in a previous report, has attracted the attention of a number of operators. Several small-size machines are being installed or are contemplated. As the prime requisite for the commercial application of the method is a workable and simple machine the laboratory machine has been further developed and may now be considered well-suited for mill operation.

The improvements incorporated in the present machine consist in a much simplified feeding device, a reduction of the steam volume behind the charge, and a reduction in size of the discharge valve and the housing of the discharge, accompanied by better impact.

Report of Investigations 3268 gives detailed results on steam consumption and efficiency under varying explosive conditions.[3]

An article was published recently which gave capacity and steaming costs as well as some typical results of differential shattering.[4]

3 Gross, John, and Wood, C. E., Explosive Shattering as a Possible Economical Method of Ore Preparation: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 11-19.

4 Dean, R. S., Gross, John, and Wood, C. E., Preparing Ore by xplosive Shattering: Eng. and Min. Jour., vol. 36, no. 6, June 1935, pp. 281-283.

2. Magnetic Properties of Minerals

The magnetic properties of minerals were only of academic interest 3 years ago when the United States Bureau of Mines began measurements on magnetite with the object of gathering data essential to a rational basis for the design of magnetic separators. The metallurgical significance of these measurements and their practical application have been discussed in previous reports, therefore only the new features developed during the fiscal year 1934-35 will be described at this time.

One of the magnetic properties commonly considered peculiar to permanent magnets is the coercive force, which measures more or less indefinitely the persistence with which remanent magnetism clings to a permanent magnet or lodestone. Coercive force in the past has had so little practical meaning that no standard method for its measurement has come into general use; it usually is defined in terms of the intercept on the H-axis of hysteresis curves. This Division has found that these intercepts of the hysteresis curves for magnetite powders were linearly related to the reciprocal of the mean diameter of the grains of the powder, showing that coercive force varies directly with mean specific surface, that is, with the surface area for unit weight or volume.[5] This discovery at once lends importance to coercive force because it provides a simple means for determining specific surface or mean grain size of magnetic powders. The astonishing feature of this relation is that it holds absolutely, as far as has been determined, whatever may be the range of sizes in a given powder. Table 1 illustrates the degree of accuracy with which the so-called "law of mixtures" holds, that is, how effectively coercive force averages grain sizes.

TABLE 1.- Law of mixtures for coercive force

With synthetic mixtures of 2 Hayden slag magnetite powders: -100 +150 mesh (180 microns mean diameter) and -325 mesh (11 microns mean diameter)

	Coercive force, Hc, oersteds	
Percentage by weight of -325-mesh magnetite	Measured	Calculated by mixture law
100	49	(49)
79.2	38.5	39.6
63.5	31	32.6
40.4	23	22.2
20.2	15	13.1
10.1	10	8.5
0	4	(4)

The coercive forces listed in table 1 were taken from the complete hysteresis curves as determined for each of the seven powders; the discrepancies between the measured and calculated values of table 1 are within the range of error of this method.

For practical utilization of the relation of coercive force to average grain sizes in a mixture more accurate values for coercive force must be used, and for this purpose a new

5 Gottschalk, V. H., Coercive Force of Magnetite Powders: Rept. of Investigations 3269, Bureau of Mines, February 1935, pp. 83-90; Physics, vol. 6, no. 4, April 1935, pp. 127-132.

method has been developed. The coercive forces listed in table 2 and plotted in figure 1 were determined by the new coercimeter and are 10 to 20 times more accurate than could have been obtained from hysteresis curves. Table 2 and figure 1 show the relation between coercive force and work input in crushing minus 8-mesh pure magnetite from the Ural Mountains by the falling-hammer method. The fact that this relation is a straight line confirms the finding of Gross and Zimmerley, of the Bureau of Mines, that Rittinger's law holds and that the work done in crushing and grinding is strictly proportional to the surface produced; or, if Rittinger's law is assumed, the straight-line relation may be taken as an independent proof that coercive force gives the true average specific surface.

TABLE 2.- Relation between coercive force and work done in crushing Ural Mountain magnetite

Sample no.	Coercive force, oersteds	Work input, kg-cm per gram
1	5.12	0.0
2	6.10	11.4
3	7.06	25.9
4	8.22	37.6
5	9.74	48.3
6	10.94	58.5
7	11.77	68.5
8	12.18	75.7
9	13.34	85.2
10	13.98	95.4
11	15.89	108.6
12	16.02	115.2
13	16.30	123.9
14	17.21	134.8
15	18.38	144.4
16	18.94	150.3

The new coercimeter is so simple in principle that a number of designs are being evolved for its use by untrained observers on the mill floor. This will enhance its utility as a rapid and convenient method for determining the theoretical amount of work required to produce a given product from a grinding machine and thus will facilitate easy determination of the efficiency of grinding.

Much of the rest of the magnetic work done during the fiscal year deals with the perfection of measuring instruments or is otherwise not of immediate metallurgical interest.[6]

6 Gottschalk, V. H., and Davis, C. W., Apparatus for Determining Magnetic Constants of Mineral Powders: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 51-65.

Gottschalk, V. H., and Wartman, F. S., Magnetization Curves for Magnetite Powders: Rept. of Investigations 3263, Bureau of Mines, February 1935, pp. 67-81.

Davis, C. W., Magnetic Properties and Orientation of Ferromagnetic Particles: Phys., vol. 6, no. 3, March 1935, pp. 96-99; A New Type of Magnetic Separator for Ore Concentration: Steel, vol. 95, no. 26, December 24, 1934, pp. 37-38; Practical Aspects of Alternating-Current Magnetic Separation: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 101-107; Magnetic Properties of Mineral Powders and Their Significance: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 91-100.

Dean, R. S., and Davis, C. W., Magnetic Concentration of Ores: Trans. Am. Inst. Min. and Met. Eng., vol. 112, 1934, pp. 509-537.

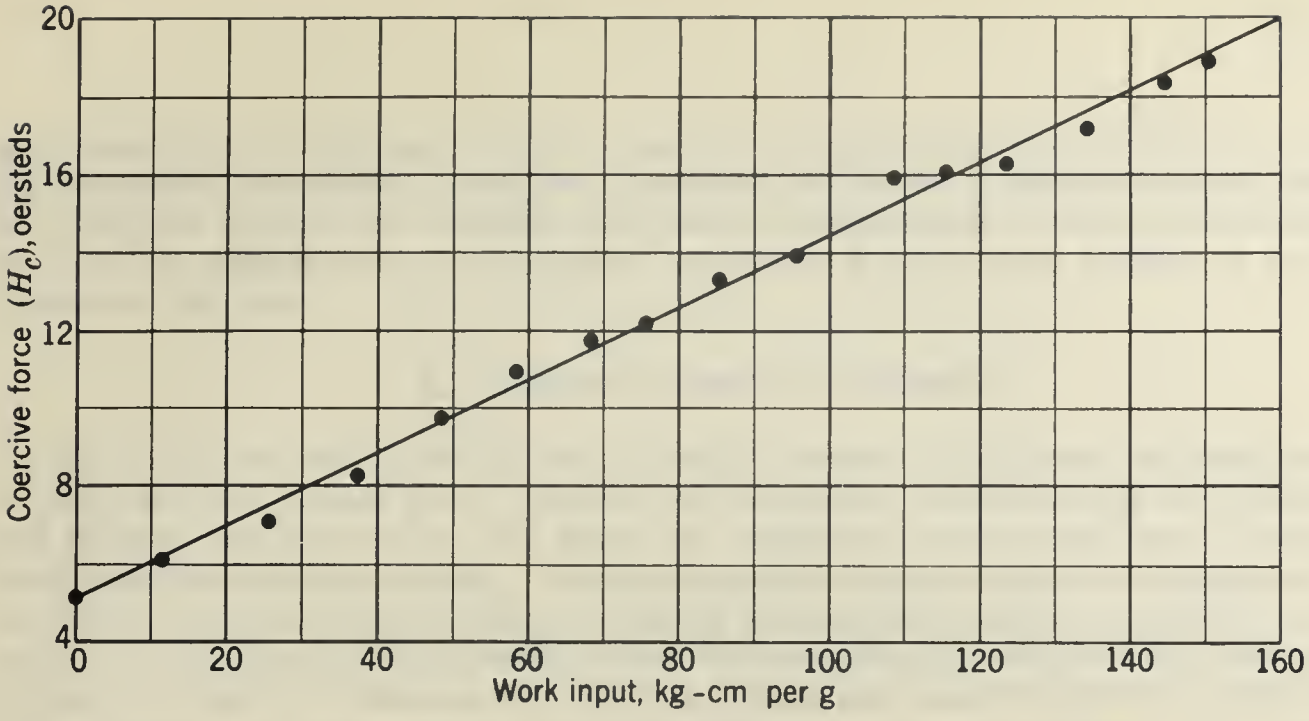


Figure 1.—Coercive force-work input relation for crushing magnetite from Ural Mountains.

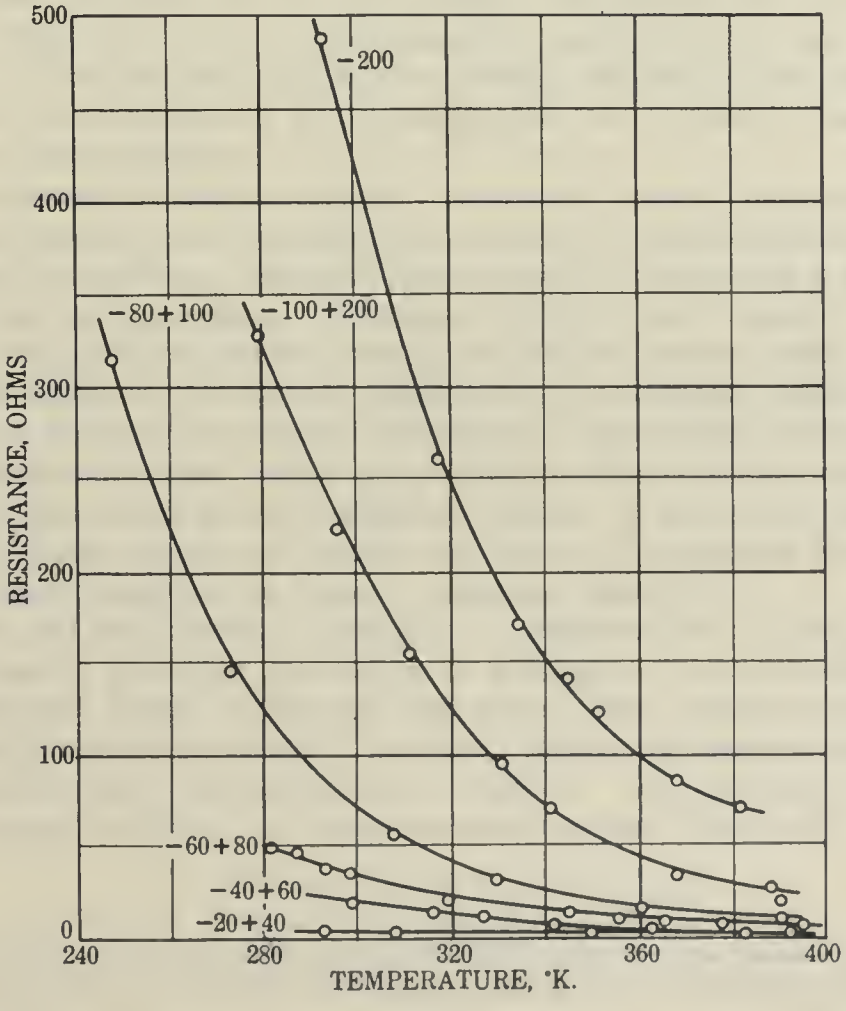


Figure 2.—Temperature variations of resistance of pastilles of sized PbS.

One unexpected practical application of one of the devices developed purely as adjuncts to the measurement of magnetic properties of powders is the new alternating-current demagnetizer[7] which was found to be amazingly efficient in demagnetizing finely ground material that could not be demagnetized by conventional methods in a pilot plant working on the beneficiation of iron ore.

3. Electrical Properties of Minerals

The first part of a study of the electrical properties of galena has been completed;[8] it shows that for massive galena crystals the temperature coefficient of electrical resistance becomes more positive as the number of translation striae (and hence the amount of mechanical deformation decreases. Fine-grained natural galena always has a negative temperature coefficient, whereas coarsely crystalline galena shows a positive temperature coefficient. The influence of grain boundaries on the temperature coefficient is shown strikingly in figure 2, where the resistance of severely pressed pastilles of carefully sized synthetic galena crystals is plotted against the temperature for the various sizes. It is to be noted that the minus 200-mesh material has a large negative temperature coefficient, whereas the minus 24 plus 40 mesh material varies but little with temperature. All of the curves follow an equation of the form $R = Ke^{A/T}$, where K and A are constants, R the resistance, T the temperature, and e the Napierian base. Parameter K varies with grain size. Quantity A is found to be quite constant for all grain sizes, indicating that the energy of binding of those electrons that give rise to the increased resistance is independent of grain size. The chief effect of decreasing grain size is simply the removal of electrons from the mobile state, perhaps by capture at the grain boundaries due to the influence of the unbalanced atomic forces at such surfaces.

Other data concerning the electrical behavior of galena, both massive and as pastilles of compressed powder, have been recorded as a result of measurements with alternating currents at various frequencies. The most interesting of these from a practical standpoint is what may be called, provisionally, "condenser effect", which seemed to be extremely sensitive to slight variations in surface conditions of the galena grains. This surface sensitivity was reminiscent of the surface sensitivity to flotation reagents and thus suggested the possibility of a radically new method of mineral separation. Experiment soon showed that surface sensitivity manifested itself also during the electrostatic attraction of the mineral particles in an alternating-current electrical field; in fact, the behavior of mineral particles on a belt moving in an a.-c. electrical field is strikingly similar to the jumping of hysteretically active minerals on an a.-c. magnetic separator.

Some idea of the electrostatic activity of minerals can be obtained from the following table. These figures represent the weight of a sample of pure mineral which jumps off the belt for a fixed belt speed, potential, and slope. The voltage between the top electrode and the grounded hot plate over which the belt passes was approximately 9,000 volts. The samples were dried at 110° to 120°C. for 30 minutes. The hot plate was on the machine in all tests, thus insuring that the minerals would be kept dry during their passage through the machine.

---

7 Davis, C. W., Some Observations on the Movement and Demagnetization of Ferromagnetic Particles in Alternating Magnetic Fields: Phys., vol. 6, June 1935, pp. 184-189; A Rapid Practical Method of Demagnetization Involving High Frequency: Nature, vol. 135, no. 3419, May 11, 1935, pp. 790-791.

8 Dean, R. S., and Koster, John, Electrical Properties of Mineral Aggregates. I. Natural and Artificial Aggregates of Crystallized Lead Sulphide: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 21-50.

TABLE 3.- Electrostatic activity of minerals

-60 +80 mesh corded-silk belt

Mineral	Weight, percent off
Bornite	91
Ilmenite	87
Cuprite	58
Arsenopyrite	43
Galena	7
Beryl	5
Chromite (Rhodesia)	0
Chromite (California)	100
Siderite	0
Rutile	0

-60 +80 mesh linen-toweling belt

Micaceous hematite	68.8
Ilmenite	46
Bornite	45
Cuprite	4
Arsenopyrite	1
Galena	0

The effect of the nap on the linen belt is seen in table 3. Much of the material became enmeshed in the fuzz and was carried over the end pulley.

Extremely good separation of magnetite from a black sand from Coquille, Oreg., can be made. An ilmenite ore from Virginia was also separable on the machine, as were synthetic mixtures of any active and inactive material listed in table 3.

There is good indication that the electrostatic activity of mineral grains is a function of the nature of grain surface. Thus, the activity of natural galena was decreased 20 to 30 percent by treating the grains with an emulsion of mineral oil and water with X-1 emulsifying agent, drying, and then washing thoroughly in C.P. benzene and redrying. California chromite agglomerated with oil and X-1 and dried thoroughly was completely inactive. Prolonged extraction with C.P. benzene followed by drying increased the electrostatic activity to about 50 percent of the activity of fresh, unoiled flotation concentrate.

The latent possibilities of this type of electrostatic separation await only further knowledge of methods for controlling the surface characteristics of mineral grains in order to warrant large-scale tests.

4. Gas Absorption on Minerals

The object of this work is to determine the amount of gas absorbed by minerals, the mechanism of absorption, and the relation of absorption to the flotability of the mineral.

The work so far has been confined to H_2S absorption on galena. Galena particles are so readily subject to surface oxidation that perfectly clean surfaces probably are not obtainable. These oxidized coatings may consist of oxides, carbonates, or sulphates, all of which are removable with a hot ammonium acetate solution. It is believed, however, that after the

removal of such coatings and before the galena can be used for gas-absorption experiments some oxidation has again taken place. Both the oxide and carbonate are converted to sulphide upon treatment with H_2S, but the water or carbon dioxide formed tends to replace the H_2S with no change in volume of gas. These coatings, therefore, are inducive to erratic results. That the treatment with ammonium acetate results in a surface change is verified by the different absorptive properties of cleaned and uncleaned galena.

A sulphate coating has no effect whatever on H_2S consumption, but any sulphate coating may mask galena to prevent absorption.

Galena samples were prepared by sieving from 100-mesh to 400-mesh and by sedimentation on the minus 400-mesh. Sedimentation was carried on to prepare samples following the sieve scale series.

Absorption tests were carried out on cleaned and uncleaned galena, and the amount of the oxide (oxide plus carbonate) and sulphate coatings was determined.

The results so far have been very erratic. There appears to be some unknown factor affecting absorption aside from the oxidized coatings. The results obtained on 50-gram charges follow:

Results of absorption tests on galena

Average size, mm	Surface, cm^2 on 50 grams	Coatings, grams: Oxide as Pb	Coatings, grams: Sulphate as $PbSO_4$	H_2S to combine with oxide, cm^3	H_2S absorbed by original galena, cm^3	H_2S absorbed less that due to oxide cm^3	H_2S absorbed by cleaned galena, cm^3	Difference between original and cleaned, cm^3
0.126	5,770	0.0154	0.0231	1.78	6.01	4.23	1.83	2.40
.089	7,755	.0176	.0304	2.04	6.80	4.76	(2.80	(1.96
							(1.87	(2.89
.063	10,560	.0224	.0319	2.59	(8.37	5.78	5.74)	1.59
					(10.61	8.02	4.89)	
.045	14.400	.0199	.0352	2.30	8.02	5.72	3.10	2.62
.0313	20,100	.0186	.0170	2.15	7.34	5.19	4.37	.82
.0221	27,550	.0481	.0140	5.56	10.04	4.48	7.85	3.37

No definite conclusion can be drawn from these results. Tentatively the following relations appear to have some ground:

1. The amount of absorption on the original is not in proportion to the oxide coating; it apparently is in direct proportion to the logarithm of the size and appears to increase with surface but not in direct proportion.

2. The amount of absorption on the original, after deduction of the H_2S used in combining with the oxide, appears to be a constant for all sizes.

3. The amount of absorption on the cleaned galena seems to be in proportion to the surface only, but the amount absorbed per unit of surface is not a constant.

4. With the exception of the finest size, there seems to be a direct relationship between the sulphate coating and the difference of absorption (between original and cleaned). This, however, does not indicate a masking effect of the sulphate coating but that this coating had a certain absorptive capacity.

Until more data are available on finer sizes any deductions can be only tentative.

The flotative effect of gassing the original and cleaned galena, of evacuating the original and cleaned galena, and of cleaning the galena was studied.

In the flotative experiments some difficulty was experienced in establishing a standard flotation. This has been possible only by determining the standard on the original galena every time a determination on a treated sample was made.

The floatability figures given in the following table are based on the amount of reagent required for complete flotation, 100 being the amount required for the original. The lower the figure, therefore, the more easily flotation is accomplished.

Flotative effect of gassing, evacuating, and cleaning galena

Average size, mm	Original galena	Original evacuated	Original gassed	Cleaned galena	Cleaned evacuated	Cleaned gassed
0.126	100	57	59	74	35	58
.089	100	64	70	61	27	68
.063	100	54	69	72	24	57
.045	100	49	37	58	21	27
.0313	100	69	42	54	24	20
.0221	100	68	45	53	27	35

Any conclusions drawn from these experiments must be accepted tentatively; it is evident, however, that the flotative property of galena is improved either by simple evacuation or by cleaning and that a combination of cleaning and evacuating is the most efficient. There does not seem to be enough evidence so far to say that gassing aids flotation. In general, improvements appear to be greater with the finer sizes.

Nothing has been published on this work as yet because the results are not conclusive.

5. Dust Settling

Considerable work has been done to show the effect of introducing different agents into a dust cloud to facilitate settling.

The problem is directed mainly toward settling flue dusts in an SO_2 atmosphere, such as that produced in an ordinary nonferrous smelting or roasting operation.

The synthetic production of such a dust cloud presents some difficulty, as flue dust once settled is difficult to redisperse. The ignition of a flue dust mixed with potassium chlorate forms a good fume, but this is quite different from a true flue dust. The method adopted to simulate true flue-dust conditions consisted in grinding a mixture of flue dust and ordinary ore minerals to an extremely fine state and blowing this into a chamber containing SO_2 atmosphere.

Many substances, both dry and liquid, were tried with and without steam, but only one - ammonia - proved encouraging.

The introduction of a small amount of ammonia results in the formation of ammonium sulphite crystals or flakes which settle rapidly and carry all dust particles with them. By this method settling may be obtained in a few minutes, whereas it requires several hours without the use of ammonia.

The most efficient method of introducing the ammonia has not yet been determined, although it seems to be best to form a layer of the ammonium sulphite in the top of the chamber.

A special application of the use of ammonia is being made on a dust from sintering machines. This dust contains a large percentage of free sulphur, and its liability to spon-

taneous combustion makes it dangerous to handle. The settling effect of ammonia is being investigated, and the effect of ammonium sulphite on combustion is to be determined.

No articles on dust settling have been published.

Three papers of a general nature have been published by the special studies section.[9]

ORE-DRESSING SECTION

The work of the ore-dressing section comprises studies of ore-dressing processes. It is concerned mainly with such basic principles as may determine the efficacy of old methods and upon which new processes may be built.

The problems selected are preferably of general rather than purely local interest; consequently, the ores used in the investigations represent types, not specific deposits.

The following outline shows the problems that have received special attention during the past year.

I. Flotation.
  - A. Oil emulsions in flotation.
  - B. Chemistry of flotation.
  - C. Flotation of nonsulphide minerals.
    1. Talc and magnesite.
    2. Stainierite.
    3. Complex molybdenum-vanadium ores.
    4. Calcite in presence of shale and quartz.
  - D. Flotation in brine.
    1. Sylvite and halite.

II. Gravity concentration.
  - A. Elutriation.
  - B. Small-scale pilot plant.
  - C. Tabling a brine pulp.
  - D. Agglomeration and tabling.
    1. Sylvite and halite.
    2. Cement raw materials.

III. Crushing and grinding.
  - A. Innovations in ball milling.
    1. Ball charge.
    2. Conical mills.
    3. Cylindrical mill with grid.
    4. Conclusions on segregated media.
    5. Concentric drums in cylindrical mills.
    6. Dry grinding.
  - B. Roll grinding.
  - C. Incidental problems.

---

9 Dean, R. S., Applied Mineral Physics: Rept. of Investigations 3268, Bureau of Mines, February 1935, pp. 5-9.
Gross, John, Summary of Investigations on Work in Crushing: Trans. Am. Inst. Min. and Met. Eng., vol. 112, 1934, pp. 116-129.
Gottschalk, V. H., Significance of Grain Surface Area: Min. and Met., Previews of Tomorrow's Metallurgy, May 1935, pp. 228-229.

Flotation

Oil Emulsions in Flotation

In the search for new flotation reagents it was found that emulsions of paraffin oils and reagents like Emulsol X-1 have merit as flotation reagents. They have enabled good separation on a synthetic lead-zinc-iron sulphide ore without difficulty. Customary depressants were used to retard the blende and pyrite while the galena was floated. The blende was later activated and floated, leaving the pyrite in the tailings. Pyrite may be easily rejected, as it shows little tendency to be floated by an oil emulsion. Tests on clean lead-iron ore gave high-grade lead concentrates without the use of a pyrite depressant.

Several tests with emulsions were made on a complex molybdenum-vanadium ore. The best results were obtained when the ore was sulphidized prior to flotation. The first results with the emulsion compared favorably with those obtained with customary sulphide flotation reagents. The emulsion of crude oil in X-1 paste makes the flotation more foolproof.

Factory-made and home-made oil emulsions were compared. Of the factory-made products Emulsol reagents 238 and 239 have been used for several ores. Both were used on synthetic sphalerite-chert, galena-dolomite, and galena-sphalerite-pyrite-dolomite ores. Reagent 238 lacks frothing properties, and considerable quantities are required for both lead and zinc cres. Even when 2 pounds per ton of ore were used lead and zinc recoveries were low. Flotation was slow, and in the tests on lead ore considerable coarse (minus 100 plus 150 mesh) galena remained in the tailings.

Emulsol reagent 239 seems satisfactory. It is a good frother, and rapid flotation of the lead ore and of the zinc ore was obtained. It does not seem particularly selective toward galena and blende when both are present, even when cyanide and zinc sulphate are used together as depressants for the blende. Lime is effective as a depressant for pyrite in the presence of reagent 239. Both 238 and 239 cause intense agglomeration of galena and sphalerite. Neither reagent will float sphalerite unless it previously has been activated with copper sulphate.

Of the home-made products two oil emulsions were investigated. One, which failed to give satisfactory results, consisted of 10 percent of Avirol 80, 7 percent of water, and 83 percent of Louisiana crude oil and was made up as a 5-percent emulsion in water. The other, prepared in the same manner, consisted of Avirol 100 and Louisiana crude oil. This emulsion is an excellent frother, especially when the froth is stabilized by a small quantity of oleic acid, and is a powerful collector for certain minerals, particularly sphalerite activated with copper sulphate. It also functions as a collector for pyrolusite and manganite and in the presence of oleic acid seems to be a good reagent for unsulphidized descloizite, vanadinite, and wulfenite, although too few tests have been made to determine whether it is superior to other reagents for this purpose.

Two features seem to be common to virtually all of the oil emulsions that have been used. They cause intense flocculation of sulphide minerals and they show a marked reluctance to float particles coarser than 150- or 100-mesh. These coarser particles are oiled so that vanning of the tailings causes the particles to film and float away from the lighter gangue minerals.

Chemistry of Flotation

This investigation is concerned with the surface conditions of minerals during the flotation process. The first study undertaken was the preferential absorption of soap and oil on sylvite. The first series of X-ray diffraction pictures was taken to disclose the nature

of these films. A picture of a rotating single crystal of KCl served to calibrate a new camera as well as act as a standard of comparison for later pictures.

In using the X-ray to study filming a potassium chloride crystal was dipped into an alcohol solution of stearic acid and the excess alcohol evaporated. An X-ray diffraction pattern showed that the stearic acid was preferentially oriented on the surface of the potassium chloride.

Pure potassium stearate was next prepared and deposited from an alcoholic solution on the surface of a potassium chloride crystal. The photogram was compared with that of powdered potassium stearate. There were indications of orientation on the surface of the potassium chloride, but the spots on the photogram, assumed to be due to the soap, did not give the exact lattice indicated by the powder photogram. A copper target was considered because it would give a denser photogram but was not used.

Owing to frequent break-downs of the X-ray tube due to leakage of gas through the tungsten-glass seals a new cathode assembly was made. It consists of a brass tube containing the filament and is cooled with transformer oil. The whole assembly is sealed to the ground joint with picein cement.

Some work was done in an attempt to prove the existence of oriented films of organic flotation reagents on single crystals of potassium chloride. Films of potassium stearate and of Avirol 80 were used, but orientation was not detected with the present technic and X-ray procedure. In an attempt to prove experimentally preferential orientation of flotation reagents on the surface of minerals X-ray photograms of galena coated with amyl xanthate and cetyl xanthate showed the patterns of the randomly oriented xanthates.

Because of repeated failures to detect preferred orientation of films on minerals a search of the literature on identification of films by X-rays was made. No mention was found of any attempt to identify flotation films by X-rays. Researches have been conducted to show the orientation of films of fatty acids on surfaces of glass, metal foil, and some minerals. Where orientation was found it was estimated that the layer of fatty acid was at least 300 molecules thick - thicker than would be expected for flotation reagents on minerals in an ordinary flotation pulp, where a monomolecular film is more probable.

During June 1935 the work was carried on in the X-ray laboratories of the University of Illinois. It had been hoped that electron diffraction apparatus would be available there, but this was not the case. Consequently, the investigation was carried on by means of X-rays.

The first series of patterns made was of the xanthates (amyl, butyl, and ethyl) on glass rods. The reagents were applied by allowing a water solution of them to dry on the glass. It was found that the xanthates became oriented so that the length of the hydrocarbon chain was perpendicular, or nearly so, to the surface of the glass. Another series of photograms of the same reagents on curved surfaces on the minerals galena, pyrite, sphalerite, and molybdenite and on rods of iron and copper showed orientation, just as on glass. There was no apparent difference in the powers of these various substances to cause orientation.

A series of photograms of minerals, taken while immersed in solutions of xanthates, showed no indications of orientation of the reagents when in the liquid state. Similarly, no effect was noticed when minerals were covered by solutions of copper sulphate and potassium cyanide.

This investigation will be continued during the coming year.

Flotation of Nonsulphide Minerals

Results of some work by the Metallurgical Division on the flotation of nonsulphide minerals has already been published.[10] The program has now been extended to include some

---

10 Clemmer, J. Bruce, and O'Meara, R. G., Flotation and Depression of Nonsulphides: Calcite, Silica and Silicates, Fluorspar, Barite, Apatite, and Tungsten Minerals: Rept. of Investigations 3239, Bureau of Mines, June 1934, pp. 9-26.

work on talc, magnesite, stainierite, wulfenite, descloizite, vanadinite, and cement raw materials.

Talc and magnesite.- A sample of talc ore was received from Johnson, Vt., for preliminary flotation testing. The ore consists of talc rather coarsely interlocked with a magnesite gangue. Small amounts of pyrite and a dark-green mineral, apparently serpentine, are associated with the talc. The discovery of magnesite was made by the Bureau. Operators had formerly called it dolomite and thought that it had no value. Now it seems that the magnesite will be worth more than the talc.

A series of products from a preliminary flotation test on waste material was sent to the Eastern Magnesia Talc Co. for analysis. A fair recovery of talc was obtained in a concentrate assaying 96 percent of talc and 0.4 percent of iron. In view of these promising results a cooperative agreement was entered into between the Bureau and this company.

The talc floats readily without the addition of any reagent, but a small amount of frother gives more rapid flotation. Butyl alcohol appears to be an ideal frother; pine oil, however, may be used in small amounts. Too much pine oil tends to float both sulphide and magnesite, but this condition may be corrected by using a small amount of sodium silicate protected by caustic soda. Sodium cyanide may be used to depress excessive pyrite.

On account of the large amount of magnesite recently identified or discovered, attempts were made to prepare by flotation magnesite tailings as well as high-grade talc concentrates. Some difficulty was experienced in depressing the magnesite and pyrite while floating the talc. Acid bichromate appeared to be superior to any other depressant, but after a few trials it was found to be unsatisfactory because it discolored the talc concentrates unless it was removed by thorough washing.

An emulsion of Emulsol X-1 paste and kerosene gave good concentrates without discoloration and assisted materially in reducing the talc content of the tailings. For example, a concentrate containing 97 percent of talc, less than 0.1 percent of iron, and representing a recovery of 70.6 percent was made, after which pyrite and fine talc were floated in a scavenger concentrate using ethyl xanthate and pine oil. The tailings contained 6.8 percent of talc and 3.11 percent of iron and represented an 80-percent magnesite recovery. The kerosene emulsion is selective, and the pyrite shows little tendency to float until most of the talc has been removed.

A further search was made for a reagent that would be an effective magnesite depressant without materially retarding the flotation of the talc. Of the reagents tested Calgon seemed to be superior. Promising results have been obtained, using the emulsion as a collector of talc and either Calgon alone or together with sodium silicate as a depressant of magnesite. Duplicate tests made with and without Calgon showed that it has merit in that it enhances selectivity and assists materially in speeding up flotation of the talc. In addition to the increase in capacity it increases the recovery of talc without a sacrifice in grade of concentrates. The cost of reagents would not be more than 20 cents per ton of ore treated.

A study of particle size in flotation of the talc showed the importance of selecting the proper mode and degree of grinding. For flotation the samples must be wet-ground in closed circuit with a classifier. Further grinding, if necessary to comply with the requirements of the industry, may be done later.

At the request of the cooperating company an 8-pound sample of magnesite and a 6-pound sample of talc were prepared by flotation from material ground in closed circuit with pebbles. These samples were sent to the company for inspection.

The flotation of this talc ore is successful and has been carried as far as it can be in the laboratory. The company must decide whether it wishes to install a pilot plant.

Stainierite.- The mineral stainierite which contains most of the cobalt in an ore from Nevada has been studied,[11] recently, and some work has been done on the use of sodium hydrosulphite in its separation by flotation.

---

11 American Mineralogist, The Mineragraphy and X-ray Analysis of Stainierite from the Swansea Mine, Goodsprings, Nev.: Vol. 20, April 1935, pp. 274-280.

An effort to find a cheaper reagent for the flotation of a cobalt ore from Goodsprings, Nev., consumed some time but met with little success. Aged and impure sodium hypophosphite was used in hot pulps, but the recovery of cobalt was low. Fresh hypophosphite may be as effective as the hydrosulphite since it is as powerful a reducer and is more desirable because it is relatively cheap.

Some extraction tests were made on black cobalt oxide and clean stainierite with aqueous solutions of ammonium hydrosulphide and sodium hydrosulphite. Both the hydrosulphide and hydrosulphite exerted a sulphidizing action on the cobalt oxides. The hydrosulphite probably reduces the Co_2O_3 to the cobaltous state, and the alkali sulphide causes formation of cobaltous sulphide which enhances flotation.

Later the home-made Avirol 100 emulsion, used to the extent of 2 pounds per ton of ore (0.2 pound Avirol 100 per ton), gave a high-grade stainierite concentrate with a high recovery. Two pounds of ammonium sulphide were used to condition the ore and a small amount of sodium silicate to depress the calcite. The emulsion is a much cheaper reagent than the sodium hyposulphite used earlier in the investigation.

Complex molybdenum-vanadium ores.- (This is a cooperative problem with the Mammoth-St. Anthony, Ltd., of Mammoth, Ariz.). A molybdenum-vanadium ore from Mammoth, Ariz., was received for study. Microscopic examination showed magnetite, hematite, goethite, pyrite, quartz, fluorite, barite, willemite, calcite, chrysocolla, wulfenite, descloizite, vanadinite, and scheelite. There is very little interlocking between any of the minerals at 48-mesh.

Detailed float-and-sink tests and microscopic examinations of the ore and products obtained by concentration tests were made. The ore itself is amenable to flotation. Soap flotation in either an acid or an alkaline circuit gave fair recovery of the vanadium and molybdenum minerals, but the concentrates were rather low-grade. The ore contains some fluor spar, barite, and calcite which float with soap reagents and contaminate the concentrate. Sulphide flotation reagents gave concentrates of somewhat better grade but with a slight sacrifice in recovery. In a typical flotation test the concentrates were 47.46 percent of lead, 7.90 percent of V_2O_3, and 16.09 percent of MoO_3. The feed was 1.06 percent of Pb, 0.22 percent of V_2O_3, and 0.36 percent of MoO_3.

In view of the success in the laboratory our efforts were transferred to the field and 3 months were spent (April to June) at Mammoth, Ariz., assisting the operators in starting a new mill. The wulfenite, vanadinite, and descloizite first were recovered by tabling and the gold by cyanidation of the gravity tailings. The recoveries for a month's operation were: 75.1 percent of the Au, 42.1 percent of the Pb, 64.1 percent of the MoO_3, and 33.7 percent of the V_2O_5. The low recovery was to be expected as the minerals aggregate in the fines and are lost to the gravity mill.

Later, as a result of laboratory findings, flotation was introduced in the mill to recover the fine mineral. Although only the primary slime was sent to flotation the results were so promising that flotation bids fair to give better recoveries than both tabling and cyanidation. Obviously three processes in a small plant are too many; one or two of them must be rejected. Evidence at hand points to straight flotation as the logical method of concentration, but many difficulties have yet to be overcome before flotation can find its proper place in the flow sheet. The weak point in flotation, as with gravity concentration, is the loss of descloizite. It floats readily in the laboratory when ground in a pebble mill, but in the plant the recovery is sometimes less than 50 percent. Iron metallics are blamed for this trouble. An investigation will be made in the laboratory to find means of counteracting the harmful effects of grinding in the iron ball mill.

This investigation will be continued during the coming year.

Samples of mill feed and flotation tailings were obtained from the concentrator of the Molybdenum Corporation of America, Questa, N. Mex. The feed sample assayed 3.84 percent of

molybdenite and the tailings 0.6 percent. The gangue is siliceous and contains small amounts of pyrite. Difficulty has been experienced at the mill in making tailings that contain less than 0.6 percent of molybdenite. Examination of the mill tailings showed the losses to be largely in the minus 400-mesh portion. Overground molybdenite is known to be difficult to recover by flotation.

Laboratory tests on the ore stage-ground to 100-mesh showed that high-grade concentrates could be produced. The grade of tailings varies considerably, depending upon the reagents employed. Lime and cyanide were used to retard the pyrite in all of the tests. With reagents similar to those used at the concentrator tailings assaying 0.4 percent of molybdenite were produced. The loss was mainly in the slime. The use of a small amount of an emulsion of crude oil and X-1 paste, together with a frother, gave tailings assaying 0.2 percent of molybdenite.

The high losses of molybdenum, constituting an irretrievable waste, are due to overgrinding and poorly selected reagents.

Calcite in presence of shale and quartz.- In the elimination of silica and alumina from limestone in cement raw materials numerous flotation tests were made on the Lehigh Valley (Pennsylvania) cement rock in an attempt to produce limestone concentrates with a SiO_2/R_2O_3 ratio of 2 or more. This would necessitate eliminating the shale without serious loss of limestone or quartz. The tests were not successful. Microscopic examination of thin sections of the ore showed that grinding to approximately 800-mesh would be necessary before such a separation could be accomplished. Consequently, attention was directed to making a clean limestone concentrate by grinding the ore to a size that would be commercially feasible and then rejecting as much of the graphitic shale and quartz as possible. Flotation tests gave promising results, indicating that a good recovery of limestone of the desired grade can be made without difficulty.

Continuation of the investigation at the company's laboratory at Northampton, Pa., is contemplated. No further work will be done on this problem by the Bureau.

Flotation in Brine

Sylvite and halite.- The potash ores of the Carlsbad (N. Mex.) district occur at a depth of about 1,000 feet and as mined consist of a mixture known as sylvinite containing about 40 parts of sylvite (KCl) and 60 parts of halite (NaCl). By careful mining the extraneous material does not amount to more than 2 or 3 percent. Such ore was used in the experimental work here reported.

The usual chemical method of recovering potash from sylvite, although efficient in the production of pure potassium chloride, requires expensive equipment and is costly to operate. The Bureau of Mines believed that the ore-dressing methods applied to other nonmetallic minerals might be adapted to the production of a grade of potash suitable for the fertilizer industry. A preliminary study showed that ore-dressing treatment was possible in a brine saturated with the constituents of the ore, and the investigation was extended through a cooperative agreement between the Potash Co. of America and the Bureau.

Grinding must adequately liberate the economic mineral from the others before physical separation can be considered. The study of polished sections of the sylvinite ore showed that the grain structure was coarse. Because of the solubility of the minerals in water kerosene was used as the wetting medium on the grinding lap. When fine grinding was completed the surface was polished with a cloth wetted with cold water. The halite and sylvite were dissolved differentially in this treatment, leaving one mineral in relief.

Early in the investigation it was appreciated that flotation might provide a satisfactory method of separating KCl from NaCl. Only slight success was attained until the

Figure 3.— Short-column elutriator developed recently by the Bureau of Mines.

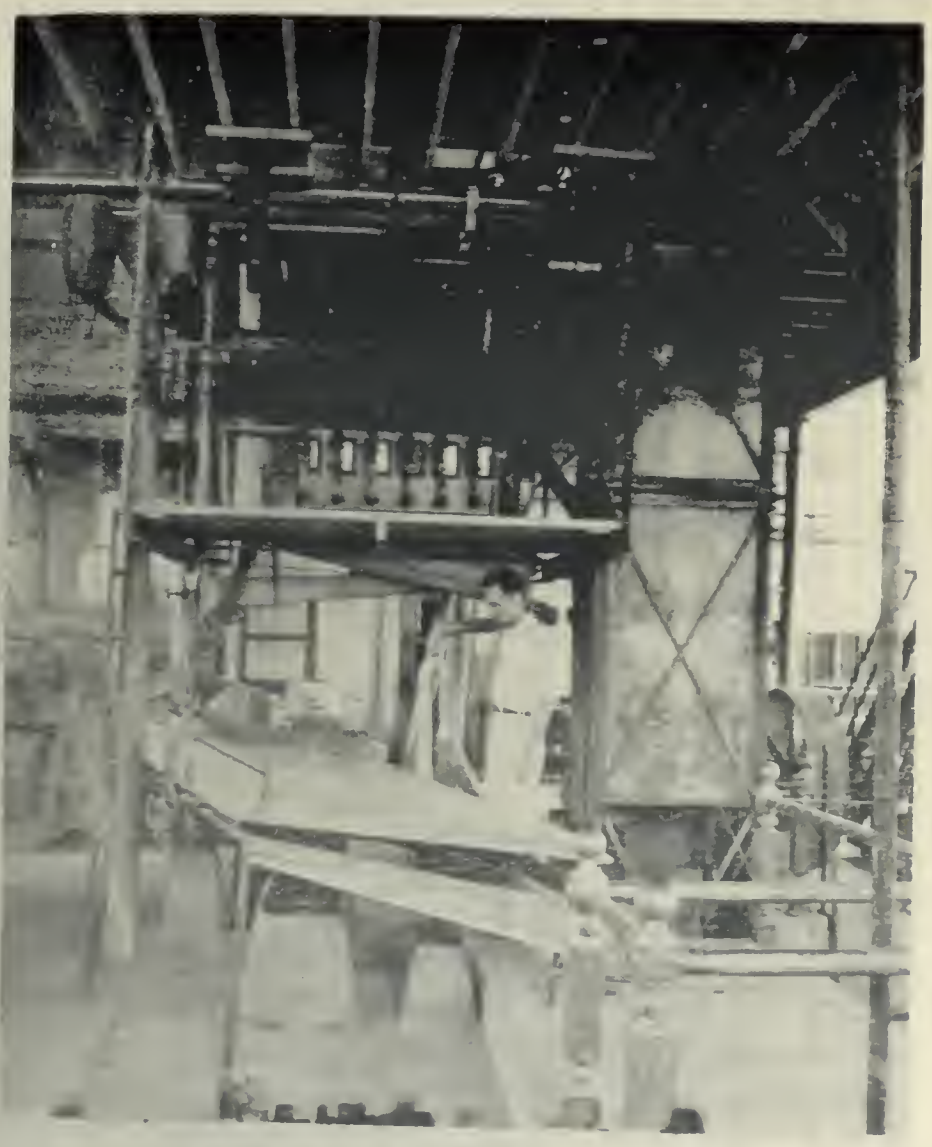


Figure 4.— Small-scale pilot plant in Metallurgy Building of Missouri School of Mines.

sulphated aliphatic alcohols were used. Excellent selective flotation of KCl, combined with low consumption of reagents, has been effected by the use of these reagents and related compounds, particularly the derivatives of those alcohols having 8 to 12 carbon atoms. Commercial forms of these reagents are available.

Although these reagents are expensive compared with ordinary flotation reagents, only small quantities are necessary. The reagent cost has been reduced to approximately 18 cents per ton of ore treated through reuse of the brine. Reused brine containing the reagent exerts a beneficial and not a harmful effect on flotation. This is fortunate, as it results in a considerable saving of reagent.

The flotation process was applicable to the ores of all grades. High-grade concentrates and low-grade tailings were produced readily. Only a small amount of a single reagent was required, and a froth with good characteristics resulted. The reagent was very selective, and flotation was exceptionally rapid. As a rule, the time required for flotation was less than 5 minutes. Slight temperature differences between ore and brine have not been found particularly detrimental. Some of the ore may be dissolved, but the grade of concentrates and tailings is not affected adversely. Since flotation is so rapid very little change in temperature was noted during the period of the tests. Crushing to 48-mesh gave entirely satisfactory results. All-flotation gives concentrates containing 85 percent of KCl with a recovery of about 97 percent; about the same grade of concentrates and recovery resulted from tabling when the fines were treated by flotation.

Gravity Concentration

Elutriation

A new elutriator has been built. (See fig. 3.) It is designed to handle 100 grams of minus 325-mesh ore which will enable large enough fractions to be prepared for chemical analysis and other examinations. The elutriator is the short-column type. This feature is an innovation, as other designers have used a long, narrow column. In the present elutriator the height has been reduced to 6 inches and the diameter increased to 5 inches.

The elutriator was used to make seven fractions ranging in size from 3.8 to 43 microns from 325-mesh material. To test the method under severe conditions the finest size - minus 3.8 microns - after precipitation, drying, and weighing was redispersed by using tannin extract and mechanical agitation in a flotation cell and was then refractionated in the elutriator into the following sizes: Minus 3.8 to 3.3 microns, minus 3.3 to 2.32 microns, and minus 2.3 microns or smaller. No particular difficulty was met in this finer sizing, and all the products were cleancut.

The difficulty usually presented in the microscopic study of such fine suspensions has been solved by rejecting the old system of flocculation, drying, and then attempting to disperse in a mounting medium. Now the elutriator overflow containing the dispersed mineral is made directly into a stiff jelly with gelatin and mounted on a slide. The gelatin insures dispersion and also prevents Brownian movement. The space between the cover glass and slide must be cemented with varnish or resin to prevent evaporation of the water in the solidified gelatin.

In the elutriation of ground tripoli from Seneca, Mo., an interesting characteristic is that the grains, which are composed of cryptocrystalline units of silica, exhibit well-rounded shapes and rarely tend to elongated or irregular shapes. In other words, the grains approximate the spherical shape assumed in Stokes' law. Hence, ground tripoli is probably unsurpassed as a material for fundamental work in elutriation. Thus far this work comprises the study and perfection of an elutriator to be used later for elutriation.

Small-Scale Pilot Plant

A classifier and table unit suitable for larger-scale continuous tests than heretofore available was built in the Metallurgy Building of the Missouri School of Mines for the use of the Bureau. (See fig. 4.) This unit consists of a six-spigot constriction plate classifier, a quarter-size table, Allen cone, dewatering tank, vibrating screen, elevator, etc., and will enable continuous tests to be made on ore samples with equipment of more nearly commercial size. The excavation of a pit 10 by 16 by 8 feet and the erection of a 32-foot elevator were required. The material was supplied by the Missouri School of Mines from its cooperative funds and the labor by the young men assigned to the station from the Professional Works Program of the Missouri Relief and Reconstruction Commission.

Tabling a Brine Pulp

Float-and-sink tests were made on sylvinite to show the extent of mineral liberation on grinding. Liberation was unsatisfactory in the plus 10-mesh fraction; consequently the ore was crushed through 10-mesh. Heavy liquids with densities of 2.14 and 2.02 were used to dissect the ore. Clean sylvite was obtained as a float product on 2.02 and clean halite as a sink in 2.14. The intermediate density from 2.02 to 2.14 contained middlings; microscopic examination showed that the majority of these grains were composed of interlocked sylvite and halite.

Relative weights of minerals in the concentrating medium are of first consideration in the examination of difficult ores for gravity concentration. A saturated brine (density, 1.23) is the only medium worthy of consideration for the concentration of sylvinite. When KCl (density, 1.99) and NaCl (density, 2.15) are immersed in this medium the relative weights are 1 to 1.21 compared to 1.16 in water. This ratio permits gravity concentration.

Closed-circuit table testing was done. (See fig. 5.) Two preliminary batches were run to develop the circulating load, which was kept constant throughout the successive runs by adding new feed. At the beginning of each run new feed was added to the circulating load for classification. The circulating load was about 2-1/2 times the amount of concentrates and tailings taken from the circuit.

The first spigot product was removed and given a mild grinding in a miniature rod mill and then returned to the classifier. This simulated plant operation with rod milling of the first spigot product in closed circuit with the classifier and retabling of all table middlings in closed circuit with the same classifier. The final table concentrates contained 82.7 percent KCl and the tailings 8.6 percent. Although the recovery was only 69.8 percent no effort was expended to make low-grade tailings because the proposed circuit included flotation after regrinding.

The flotation of the reground table tailings along with the classifier overflow was successful. Flotation made concentrates containing 89.2 percent of KCl and tailings containing 1.8 percent of KCl. A summary of tabling and flotation shows a recovery of 97.3 percent of the KCl in concentrates of 84.9 percent of KCl grade.

Agglomeration and Tabling

Sylvite and halite.- Many reagents were tested in a search for those which would permit selective filming or agglomeration of KCl preparatory to tabling. From the point of cost and efficacy Emulsol X-1 was the most suitable for agglomeration and filming. Instead of being "salted out" in brine, as are the ordinary soaps, these substances form finely dispersed emulsions with the brine.

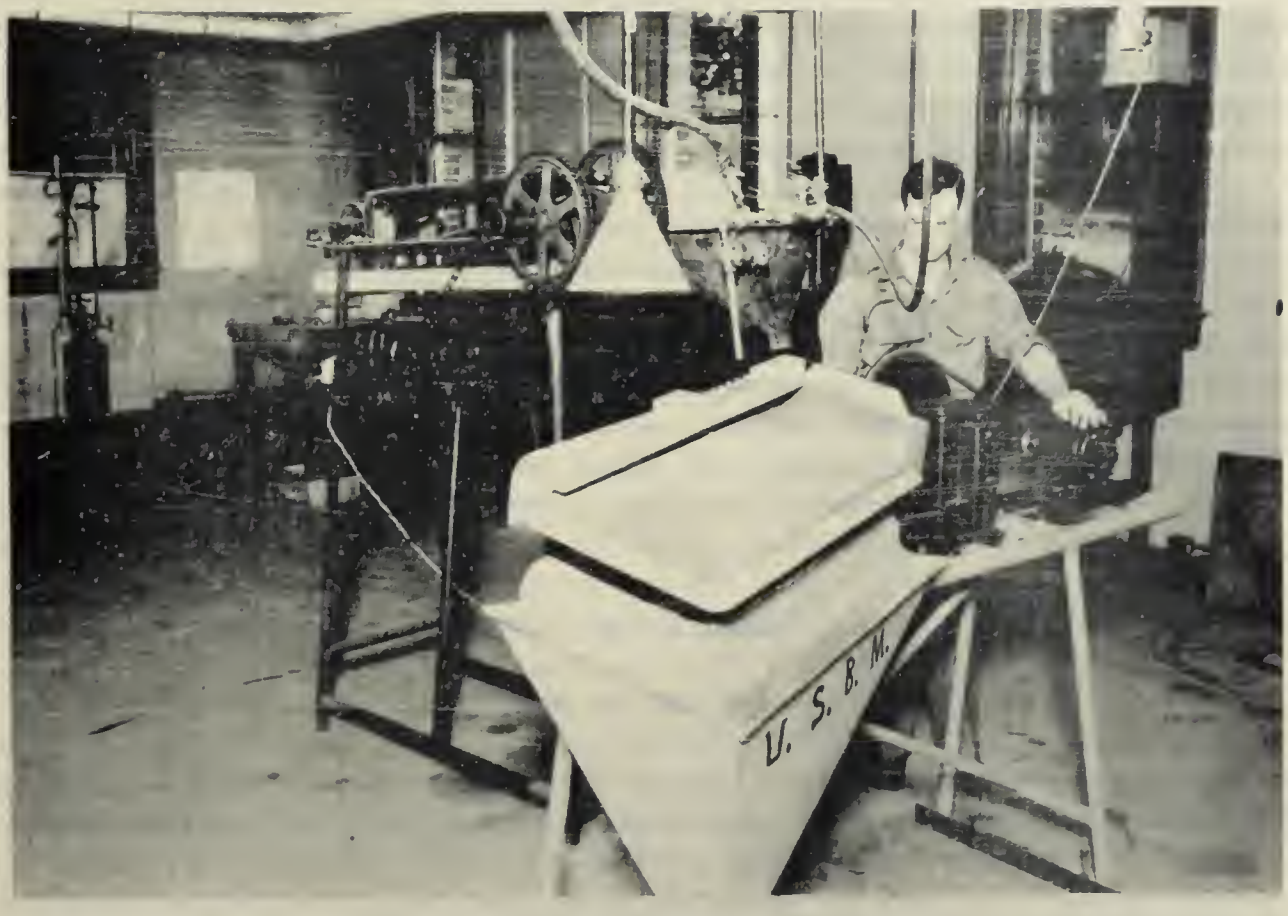


Figure 5.— Diagonal-deck concentrating table used in potash work.

Agglomeration and tabling have a distinct advantage over straight tabling in that high-grade concentrates and low-grade tailings are produced with a minimum of middlings. These middlings are readily re-treated.

The results of the work on this problem are summarized elsewhere.[12]

Cement raw materials.- (This is a cooperative investigation with the Universal Atlas Cement Co.). During May 1934 a limestone containing free chert was received from the Hannibal (Mo.) plant of this company. The company wished to know if the limestone could be eliminated by ore-dressing methods. The limestone grains had a specific gravity of 2.68 and the chart grains 2.59. Jig tests in water and in brine gave partial separation; the separation probably was due less to the difference in specific gravity than to the markedly flattened character of the chert particles. Preliminary agglomeration and table tests on 10- to 28-mesh limestone containing approximately 12 percent of insoluble gave a limestone concentrate assaying 1.6 percent of insoluble and represented a recovery of 75 percent. The reagents used were crude sodium oleate and Bartlesville crude oil.

In view of the preliminary examination the company decided to enter into a cooperative agreement to pursue work on this problem. Intensive work was done during the first quarter which showed definitely that the cherty-lime-stone ores may be concentrated successfully by agglomeration and tabling to yield a clean limestone concentrate suitable in every respect for cement manufacture. Several tests were made to demonstrate the process to the cooperating company's officials who visited the station. The company is now considering applying the process to their Hannibal (Mo.) plant should the economics of the process and business conditions warrant the changes necessary in the present plant set-up. Details of this investigation are published elsewhere.[13]

Crushing and Grinding

Innovations in Ball Milling

Ball charge.- Many years ago E. W. Davis assumed that the rate of wear of the different sizes of balls in a ball mill was directly proportional to the weight of each ball, and he evolved a formula from which to calculate a balanced charge. Operators have used this formula when purchasing balls for a new mill or reloading an old one that had been emptied for repair. The formula required that the largest ball size and the size to be rejected should be determined, and after that the amount of the other sizes was set. Stress was laid upon the coarsest size, and many writers have made their contribution by reporting ratio of particle size to optimum ball size to facilitate use of the formula.

The inadequacy of the formula and the futility of extensive experimentation for ratio determinations is at once obvious when one realizes that the formula did not consider the grinding rate of the finer sizes. To be sure, a few operators who were grinding to very fine sizes altered the make-up charge by using some small balls with the big ones, but this practice has been somewhat haphazard.

Today operators have a keener sense of the relatively large amount of work required to finish the finest sizes, so the insufficiency of the formula is readily seen. It would have been fortunate if the formula had been devised to attract more attention to the finer particles. The literature is first-class from the standpoint of ball wear, but it gives very

---

12 Coghill, Will H., DeVaney, F. D., Clemmer, J. B., and Cooke, S. R. B., Concentration of the Potash Ores of Carlsbad, N. Mex., by Ore-Dressing Methods: Rept. of Investigations 3271, Bureau of Mines, February 1935, 13 pp.

13 Diener, F. P., Cooke, S. R. B., and Clemmer, J. B., Beneficiating Cement Raw Materials by Agglomeration and Tabling: Rept. of Investigations 3247, Bureau of Mines, March 1935, 6 pp.

little about a balance for the best grinding of all particle sizes extending throughout the length of the mill. Research has accepted this matter in a spirit of submission.

That the coarse particles must be crushed is not denied, else no fine material would accumulate; emphasis, however, is laid on the fact that when crushing to 200-mesh the operator should select balls for crushing, say, from 100- to 200-mesh, or when crushing to 65-mesh he should judiciously load the mill for crushing from 48- to 65-mesh. If this practice were followed the grinding would be much better, and the circulating load would be relieved of the large amount of nearly finished size. In its stead would be some coarser material from which the classifier could more easily remove the finished size.

Published tests have been supplemented by detailed information on the optimum size of balls for grinding sized ore. Figures have been obtained which show conclusively which size of ball is the most efficient in crushing certain sizes of dolomite and chert. In crushing material of the size range 48- to 65-mesh the increased efficiency of small balls has been strikingly demonstrated. As regards capacity and efficiency small balls 5/8 inch in diameter were more than twice as efficient as 3-inch balls.

The old formula was therefore set aside temporarily for one that included all the sizes of particles and the size of medium that was suited for each size of particle down to the finishing size. This formula gave a ball charge that differed widely from the old ones; a great many small balls were required. Comparative grinding tests gave more effective grinding by the new charge.

Conical mills.- The discussion about the ball charge points to making it up in accord with requirements at the finish as well as at the beginning of the grind. Should the sizes be segregated or should they be mixed? In the metallics industry they are segregated only when mills are placed in series and succeedingly smaller ball sizes are used from the first to last mill in the series. In the cement indstry dividers or grids are used to divide the very long mills into sections each of which has the appropriate size of medium.

With the knowledge that the cone of a conical mill functions like a grid in segregating the balls with respect to size conical mills were built and tested at the Rolla station. The first one was only 3 feet long; it proved that a steep cone is not required for grading the balls. A taper of 2 inches to the foot was ample to segregate the largest balls in the big end and the smallest balls in the small end.

Grinding tests in this mill with a ball charge proportioned in accord with recent experimental results were compared with the old cylindrical mill loaded with the old-style ball charge. A decided advantage was gained by the newer practice; it resulted in 58 percent more grinding for a unit of power.

To determine whether the improvement was due to the conical mill or to the new ball charge or both, a larger mill was built. It was 6 feet long and had the same taper as the first one. The big end was 2 feet in diameter and the small end 1 foot in diameter. The power to segregate the balls was put to the severest test; the small balls were placed in the big end and the big ones in the small end, and after a few minutes of operation their positions were reversed. Having given positive evidence of the segregating effect, the mill was ready for use.

Grinding tests with several combinations of mills and charges led to the conclusion that the advantages which had been gained were due more to the new ball charge than to the new design of mill. It was difficult to show that the conical mill had advantage over the cylindrical mill. In fact, the 6-foot conical mill had a disadvantage; it induced the ball load to drift to the big end and pile up there to such an extent that it passed through the feed entrance into the scoop. A grid had to be placed on the feed opening to retain the charge.

Considering the large amount of work required to grind the fine sizes, it became apparent that a long, conical mill with its smallest diameter where the greatest amount of work is

done is about as near to being the antithesis of a good design as anything that could be contrived. Hence, a change was made to a cylindrical mill lined with a series of truncated cones.

A 2- by 3-foot cylindrical mill was lined to give the conical effect, but instead of having one cone it had three truncated cones end to end and apexing in the same direction. Any cylindrical mill may be so lined by using tapered liners. If the liners are 2 feet long, a 12-foot mill would have six truncated cones end to end. The mill would have the same capacity at the discharge end as at the feed end. In some ways, at least, this would be an advantage over the long cone. The mill with truncated cones proved to be as good a sizing device as the long, single cone, but when compared with the old cylindrical mill its advantage as a grinder was not marked.

Cylindrical mill with grid.- Finally a grid was placed 1 foot from the feed end of a 2- by 3-foot cylindrical mill to keep the ball sizes segregated. Balls 2-1/2 to 1 inch in diameter were placed in the feed-end section and 3/4-inch balls in the discharge end. The grinding was scarcely better than if the grid had been absent. The grind would have functioned more advantageously if the feed had been coarser and the finishing finer.

Conclusions on segregated media.- The results obtained with the conical mills and with the cylindrical mill containing a grid show the difficulty of finding a substitute for the plain cylindrical mill.

The validity of graded sizes of balls to grind the ore in steps cannot be denied, but more research is required. Ample provisions must be made for a circulating load in each step.

In the conical mills or in the grid mill as used in these tests the rate of feed could not be properly set. If the feed were a little too fast some of the coarse particles would pass the zone intended to grind them and would likely continue without being ground. Again, if the feed were too slow energy would be wasted by making the fine particles remain too long with the coarse medium; overgrinding and inefficiency would result.

The new ball charge showed about the same superiority in the plain cylindrical mill as in the others.

Concentric drums in cylindrical mills.- As a means of increasing efficiency other modifications of the ball mill have been suggested. For example, to obtain greater impact a designer of grinding equipment conceived the idea of placing a cylindrical drum in the ball mill with its axis coincident with the axis of the mill. Each of three drums have been tried in a 2- by 3-foot mill; they were 8-3/4, 11, and 12-1/2 inches in diameter.

According to the statements of the inventor the balls are thrown from the top of the drum and meet the heap below by impact. Doubtless the grinding is by impact as claimed, but a thorough examination shows that at different speeds with drums of various sizes and with various ball loads the products from the grinds are quite similar to those made without the drum. The ultimate effect of the drum was to reduce the required power, and the loss in grinding was proportional. Hence, the drum reduces capacity without increasing efficiency.

Dry grinding.- The discussion thus far relates to wet grinding. Recently, however, a study of dry grinding has been made; the tests showed that with the same amount of ore in the mill and at the same speed the capacity of the mill grinding wet is about 20 percent higher than when grinding dry. The grinding efficiency of the wet charge also is about 20 percent higher than the dry charge with ore loads proportional to those usually found in practice. Screen analyses of dry-ground and wet-ground products having the same percentage of minus 200-mesh material were examined to see which method gave the best selective grinding. Selective grinding was found to depend on the fixed and induced variables. The results have been set forth in extensive tables.

In a further study of dry grinding the Universal Atlas Cement Co. in its cooperative agreement with the station has introduced an important problem in the dry grinding of cement clinker. The company has thirty-eight 7- by 22-foot pebble mills converted to ball mills. On account of the light construction and correspondingly small motors the ball load is limited to 18 percent of the mill volume. The efficiency of such a small volume was questioned. Investigation disclosed that the overflow type of mill is inefficient when the ball volume is so small, but if the mills are of the quick-discharge or low-pulp-level type and maintain a low pulp level in accordance with the manufacturers' claims grinding will be efficient and proportional to the power expended. Earlier work had already shown to what extent an excessive amount of ore in the mill reduced efficiency. The ratio must be kept within a limited range. The high ratio of ore to balls is the reason for the overflow type being so inefficient with a small load.

Roll Grinding

Roll-grinding tests giving particular attention to the effect of the spring tension were made with a set of 10- by 6-inch rolls. Heavy railroad-car springs to replace the lightweight factory springs were obtained and calibrated. Crushing tests were made with spring pressers ranging from 500 to 7,000 pounds for each of the two springs.

By trial and error the amount of feed was made proportional to the horsepower - a difficult task requiring much time and skill. By this method of setting a comparison of the sizing analyses of the products showed the relative efficiencies which were all within the range of those obtained in ball milling. This investigation showed that if ball mills are as inefficient as some theorists have indicated the rolls also must likewise be condemned.

Incidental Problems

In the early work in ball milling in which the laboratory mills were compared with the mills in concentrators the conclusion was reached that the performances were identical. Later, when the report was being drafted it was realized that notes in a form to convince the reader were not available. Accordingly, a 6- by 4-foot ball mill was tested, and a feed sample for laboratory tests was taken by intercepting the regular flow to the mill. When this sample was ground in the 19- by 36-foot laboratory mill it was found that the products from the plant and laboratory tests were identical in type, but the capacity of the big mill was deficient about 8 percent. Either this discrepancy is due to fundamental differences or to an error in determining tonnage or power in the plant. The performance of the 24- by 24-foot and 19- by 36-foot laboratory mills have been found to be identical.

Cooperation with one of the talc-mining companies disclosed that talc miners are unfamiliar with wet closed-circuit grinding. Therefore, the work of grinding talc was taken up in the laboratory as a plant problem. This work illustrates how the present laboratory can be used in the future to outline new problems. The feed was through 8-mesh and the finished overflow through 48-mesh. Flint pebbles were used in grinding. The net power was 5.1 hp.-hr. per ton. About one-third has to be added to net power for gross motor input. Hence, the motor input would be 6.8 hp.-hr. per ton. A more extensive study would require elutriation of the fines.

The advantages of using ball mills of larger diameters and those having improved bearings have been recorded recently.[14]

---

14 Coghill, W. H., and DeVaney, F. D., Advantage of Ball (Rod) Mills of Larger Diameters and Advantage of Improving Bearings: Trans. Am. Inst. Min. and Met. Eng., vol. 112, 1935, pp. 79-87.

IRON AND STEEL SECTION

This section studies various phases of processes incident to the manufacture of iron and steel, from crude ore to the finished metal, with the purpose of improving present practice and present products. It is also interested in the development of new processes which may simplify or improve the production of superior iron and steel.

The work of the past year has followed two lines: (1) Studies in direct production of steel and (2) blast-furnace investigations. The former was concerned with the utilization of a re-formed natural gas-air mixture to produce sponge iron, which in turn was melted. The latter included studies of size preparation of iron ores, desulphurization, and methods of charging and gas distribution in the blast furnace.

Two papers on the direct production of iron and steel from ores were published by the section during the past year.[15]

Blast-Furnace Investigations

Size Preparation of Iron Ores

It is generally recognized that hard, dense ores - often called refractory ores - can be smelted with less fuel when crushed to a size that will permit substantially complete reduction in the upper part of the blast furnace; a number of plants have made large savings in coke by finer crushing of dense ores. Development in size preparation of iron ore is being retarded because no simple methods have been developed by which operators can determine the relative reducibility and proper crushing of their ores.

The apparent density, true density, porosity, and relative reducibility were determined for 120 specimens of 35 iron ores from Minnesota, Michigan, Alabama, New Mexico, Wyoming, and Utah. The relative reducibility under a set of standard conditions ranged from about 6 for dehydrated porous limonites to 1 for dense hematites approaching zero porosity.

A relation involving porosity, relative reducibility, and degree of crushing of an ore which permits the determination of the proper degree of crushing for hematites or altered magnetites has been established by testing a wide variety of ores to find the degree of crushing that practice and furnace surveys made by the Bureau have proved to be satisfactory.

Methods of Charging and Gas Distribution

Methods advocated by the Bureau of Mines for determining the gas-flow characteristics of blast furnaces and for correcting undesirable conditions by changes in methods of filling were applied to two commercial furnaces. Correction of excessive channeling of the gas through the center, as well as along the walls, by varying the weight of ore per charge decreased the coke required by 250 to 350 pounds per ton of iron.

Desulphurization Studies

Desulphurization in the blast furnace has been assumed (1) to occur across the interface between the metal bath and the slag bath and (2) to result from drops of metal descending through the slag bath.

---

15 Dean, R. S., Barrett, E. P., and Pierson, Calvin, Some Observations on Sponge Iron and the Properties of the Direct Steel Made from It: Am. Inst. Min. and Met. Eng., Metals Technology, Tech. Pub. 592, vol. 2, no. 1, 1935.
Dean, R. S., Present Status of Direct Production of Iron and Steel from Ores: Min. and Met., Apr. 1, 1935, pp. 185-186.

Owing to the inherent slowness of diffusion processes, CaS, one of the reaction products tends to accumulate at the interface between quiet slag and metal baths, as shown in figure 6,A and B, sulphur prints in which the dark lines indicate the high sulphide concentration at the interface. The reaction in which the metal drops descend through a slag bath differs from that of the quiet baths of metal and slag in that the relative motion of the former prevents the formation of a high sulphide concentration at the surface of the drops and therefore favors desulphurization.

Attempts to evaluate the relative efficacies of the two means of desulphurization by laboratory tes s led to the following conclusions: (1) Desulphurization in quiet baths of slag and metal is faster than might be expected in view of the limiting effect of diffusion, and (2) descen of metal drops is too rapid to utilize fully the washing effect of the moving drops.

To account for the former conclusion one may consider the accepted reactions:

$$FeS + CaO = CaS + FeO$$
$$FeO + C = CO + Fe$$

FeO is liberated at the interface where C, dissolved in the metal bath, is available to produce CO. Accumulation of CO results in a gas bubble which eventually rises through the slag bath. An immediate effect is a distortion of the interface film containing the high sulphide concentration and consequent movement of S into the slag bath. This is shown in figure 6,C, a sulphur print in which a bubble of CO in slag is outlined by a high sulphide film. Furthermore, the formation of metal drops in the slag bodies of quiet-bath tests indicated the removal of metal from the metal bath by some means. This also was traced to the formation of CO at the interface, the rising bubbles lifting and transporting minute drops of metal through the slag. This buoyant effect of gas bubbles in the slag is shown in figure 6,D, in which two of a number of beads of metal partly fill the gas pockets in the slag adjacent to them. Slags melted over metal baths containing no sulphur are devoid of metal beads or drops (fig. 6,E); those melted over metal baths containing sulphur were always found to contain metal (fig. 6,F).

Sulphur prints of cross sections of slag and metal always showed the metal held in the body of the slag to be virtually sulphur-free. Figure 6,G, is a sulphur print made from the test piece shown in figure 6,H. Tests show that drops of metal like those in figure 6,H, fall through slag too rapidly for efficient desulphurization. Furthermore, in this case no gas pockets are to be seen above them. It is therefore assumed that they are on their return trip to the metal bath, the result of the coalescence of innumerable minute droplets, attached to CO bubbles, which were completely desulphurized on their upward trips.

This phenomenon must modify both of the simple conceptions of the desulphurization process mentioned at the outset.

Many tests have been made to determine the relation between the chemical composition of slags and their relative desulphurizing powers. Broad generalizations in regard to this relationship are impractical. For example, it is generally stated that an increase in basicity results in an improvement in the desulphurizing character of a slag. It has been found, however, that for a limited range of composition CaO and MgO are equal in their effects on desulphurization when computed on a molecular basis. In another limited range their effects are equal when composition is computed on a weight basis. A range of composition has been found where the addition of CaO decreases desulphurization, whereas the addition of MgO improves it.

A survey is under way in which relative desulphurizing values for all likely slag compositions will be determined and assembled in tables and diagrams so that the effect of slag

Figure 6.– Desulphurization in the blast furnace.

composition on desulphurization can be seen readily. This survey, as at present projected, will cover all compositions within the following range:

SiO_2 - 30 to 50 percent, by 2-percent intervals.
Al_2O_3 - 5 to 25 percent, by 2-percent intervals.
CaO - 35 to 55 percent, by 2-percent intervals.
MgO - 0 to 20 percent, by 5-percent intervals.

These ranges may be broadened as results may indicate the desirability of such an extension. To data desulphurization values have been determined on slags of 186 compositions. These data have been charted on triaxial diagrams showing SiO_2, Al_2O_3, and CaO contents, each diagram being a plane representing one MgO content. Diagrams for zero and for 5 percent magnesia have been completed for the ranges of SiO_2, Al_2O_3, and CaO given. The desulphurizing values are recorded on the diagrams for the respective slag compositions, and lines are drawn to connect points of equal desulphurizing powers - isodesulphurization lines.

It is expected that when completed this data will find valuable application in blast-furnace practice. It will also have a theoretical interest when used with data already available in showing the relationship between desulphurization and viscosity.

A progress report showing the relation of desulphurization studies to changes in the ratio of silica to alumina in iron ores has been published.[16]

Precious-Metals Section

The program of the precious metals section covers studies of various processes incident to the recovery of precious metals from their ores. The work during the past year has included the study of processes as a step toward their extension or improvement and the study of ores to determine the preferred method for their treatment.

Studies in the first group involve investigations relating to flotation, amalgamation, and cyanidation while those in the second, studies of coated gold, gold in pyrite, and gold in miscellaneous ores.

Flotation of Free Gold and Silver Minerals

The present work on the flotation of gold is an extension of that previously reported.[17]

Experiments showed that sodium sulphide is a depressant for free gold and for the silver minerals argentite and cerargyrite. This is a point of considerable significance in the treatment of oxidized ores containing gold and silver in which the general practice uses sodium sulphide to increase the floatability of certain oxidized minerals.

Under a certain set of conditions 81.2 percent of the gold in Idaho-Maryland ore floated but only 2 percent floated in the presence of sodium sulphide equivalent to 2 pounds per ton. The depressant effect on Druid ore, however, is not strong enough to permit the satisfactory separation of the gold from the pyrite.

Long conditioning greatly diminishes the depressant effect of sodium sulphide. When Idaho-Maryland ore was conditioned for some hours in the presence of sodium sulphide 68 percent of the gold floated.

---

16 Joseph, T. L., and Holbrook, W. F., Silica and Alumina in Iron Ores: Blast Furnace and Steel Plant, May 1935, pp. 313-316.

17 Leaver, E. S., Woolf, J. A., Flotation of Metallic Gold (Relation of Particle Size to Floatability): American Cyanamid Co., Ore Dressing Notes No. 2, September 1934.

Sodium sulphide also acts as a depressant for coated gold in spite of the fact that the microscope shows that the surfaces of such particles are tarnished by the sulphide. Thus, when the Buckskin mine used sodium sulphide to float the tarnished pyrite with which the gold was supposedly associated and amyl zanthate plus aerofloat 15 as collectors, the tailings contained 0.10 ounce of gold. By simply omitting the sodium sulphide the gold content of the tailings dropped to 0.005 ounce.

Flotation of Silver Minerals

In addition to the depressant action of sodium sulphide certain other phenomena have been observed concerning the action of synthetic silver "ores" in flotation. Concentrations of lime up to 1 pound per ton have no noticeable effect, but when a lime concentration of 3 pounds per ton has been reached much silver is depressed during flotation. When cerargyrite has been ground with silica it may be floated readily to produce a rich concentrate. When clayey or slimy material is added the recovery is still good, but the grade of concentrate drops. Similar results were noted with synthetic argentite "ores"; the discovery was made that the detrimental action of clay could be partly overcome by the addition of starch.

Flotation in Acid Pulp

A mixed amyl xanthate-aerofloat 15 collector works differently in acid than in alkaline circuits. A highly siliceous Eureka-Lily ore works better in the former, but Sunbeam-Buckskin ore and Pine Grove ore are more amenable to treatment in the latter type of circuit, the Pine Grove ore yielding 93 percent of its gold in alkaline circuit and only 43 percent when acid was present. It is thought that the treatment is more concerned with the presence of H_2S than is the pH of the pulp so that an acid circuit is not to be recommended when conditions favoring the formation of H_2S are present.

Use of Fatty Acids as Collectors for Coated Gold

It has been found difficult to float more than 65 to 70 percent of the gold and silver in the Eureka Lily ores, which contain some 92 percent of siliceous material. Virtually no slime is found by very fine grinding. If all the bases were floated the resulting concentrate would be very acceptable to the smelters. The gold lost in the tailings an iron-coated variety, so that this ore was chosen for extended work on floating coated gold. The gold is very fine, none being coarser than 150-mesh, and about 10.01 ounces of gold per ton are insoluble in aqua regia. Apparently some of the gold is "locked up" in silica.

After floating the clean gold with standard collectors such as xanthates it was found that the addition of small amounts of fatty acids floated 20 percent additional gold. An attempt to improve the action by the addition of various modifiers that might form insoluble precipitates on the coated gold surfaces was not successful.

Fatty acids are not very effective in the flotation of clean gold, but they exert a preferential action on lime and iron oxides. The recoveries in the latter, however, are not especially good. The addition of metallic salts improves the grade of the gold concentrate but does not increase recovery.

Flotation of Scheelite

Most of the tungsten produced in the United States is made from scheelite ores. The mineral occurs in metamorphic contact deposits and thus has approximately the same accompany-

ing gangue constituents regardless of the locality in which found. Mills are now making marketable concentrates by gravity methods following careful stage crushing and hesitate to change to small flotation circuits for fear that the finer-size concentrate would not be marketable. They only wish to float the slimed material in tailings where the loss of scheelite is known to be high. Tests in one mill showed that 95 percent of the total WO_3 lost was minus 200-mesh.

It was found possible to float almost all the scheelite as a low-grade material containing some 4 percent of WO_3, but when the work was done in the mill it was learned that the hardness of the mill water was a factor of great importance. The water contained as much as 1,680 parts per million of sulphates, largely of magnesia and calcium, so that it was found necessary to take thickener slime to a separate circuit to dilute it with fresh well water and to use a batch system to float the scheelite, with a reasonable amount of reagents.

Hard water consumes reagents and causes gangue to be carried into the concentrate. Sodium carbonate and sodium silicate counteract these actions, but in the last mill test 12 pounds of the former and 2.5 pounds of the latter were needed to properly condition the feed. Under these conditions 84.9 percent of the total WO_3 was floated in a 19.25-percent WO_3 concentrate. This material, which is diluted by decomposed calcium carbonate and other products of rock decomposition, will be investigated further.

Amalgamation by Fine Grinding

Amalgamation, because of its simplicity and low plant cost, is often selected as a process for the recovery of gold when the ores are amenable to treatment by this method. Extension of the process would be particularly beneficial to the small operator; results indicate that even large plants might profitably recover more gold by amalgamation.

Tests were made to determine the effect of very intimate contact between mercury and ore and of various impurities upon the amount of gold amalgamated.

It was found that the presence of the usual sulphides, oxides, and carbonates of copper and lead have little effect on the loss of mercury incident to fine grinding. Realgar and stibmite, however, cause finely ground mercury to be lost. Pyrrhotite and marcasite were found to cause more loss than did pyrite, but carbonate schists have no effect on the percentage of gold removed when amalgamation follows fine grinding.

Two methods were used to disperse finely divided mercury through various ores. In one the ore and mercury were ground together. In the other finely dispersed mercury was added to ground ore. Mercury was dispersed by precipitating it from a solution of mercury chloride with metallic zinc and by condensing mercury vapor on cold ore. Plate amalgamation was more effective on samples containing dispersed mercury than on others. By using dispersed mercury the amount of gold amalgamated from one ore was increased from 65 to 85 percent. A disadvantage of using dispersed mercury is the difficulty of recovering floured mercury in a reusable form. If floated the concentrate would have to be treated before quicksilver could be used.

The best amalgamation results were obtained by grinding the ore fine enough to free all the gold and then adding the pulp to a slowly revolving drum amalgamator. The loss of mercury in this procedure is small. An example of the efficiency of the process in recovering gold is furnished by Idaho-Maryland ore which yielded 91.8 percent of its gold when ground so that only 1.2 percent of the material remained on 200-mesh. Finer grinding was detrimental. The loss of mercury was 0.2 to 0.5 pound per ton. Other tests on Montezuma-Apex ore and on Homestake ore substantiate the belief that there is an optimum size at which each ore should be amalgamated.

Use of Gold Cloth

Gold cloth is regaining favor as a means of recovering gold lost in flotation by gravity processes or amalgamation. It is also surprisingly effective when used by itself. Three well-known ores from Idaho-Maryland, Moctezuma-Apex, and the Homestake mines yielded 86.2, 82.7, and 74.5 percent of their gold as 3.9, 0.93, and 1.34 ounce concentrate, respectively, by the direct use of gold cloth. Adding the gold recovered as middling products gives a recovery of 91.2, 85.5, and 87.2 percent of the gold as a low-grade concentrate. Unfortunately, ores that are difficult to treat usually leave but little of their gold on this cloth. Tallapoosa district ore gives up only 25 to 30 percent of its gold, and Proskey ore gives up very little. The Weepah ore yields 83 percent of its gold content, but the concentrate only 0.47 ounce per ton. It would seem that gold cloth might be an excellent supplement to the other ore-dressing methods now in common use.

Cyaniding Gold Flotation Concentrate

Shipment of gold as bullion has many advantages, particularly for the small mine owner in isolated districts. When this is not feasible it is an advantage to ship a concentrate of as high grade as possible which contains preferably those ingredients that interfere with cyaniding the flotation tailings or middlings.

An extraction of 94.5 percent of the gold in the table concentrate from the Forest Hills mine of the Mother Lode district of California can be made by cyanidation. Roasting this concentrate makes the gold more insoluble.

The Druid mine and mines of the Kansas-Burroughs district in Colorado are now rounding into production. The gold can be floated but then is difficult to dissolve in cyanide solution even after roasting and fine grinding. Careful manipulation, however, permitted a first flotation concentrate worth $190 per ton and a second worth about $25 to be made. Some 96 percent of the gold in the Kansas-Burroughs ore was floated as a low-grade concentrate. By recleaning the concentrates a product of shipping grade perhaps can be made and the middling product then treated by other methods.

Coated Gold

In addition to the work already reported on the flotation of coated gold a further study of the nature of coated gold particles and methods for their treatment has been started.

Coated gold particles from the Mercur district were examined at magnifications up to 1,250 diameters. The particles had reddish, dirty-gray, or black surfaces with uneven striations; these surfaces were of a pebbly nature so that coatings adhered tenaciously. Carbonaceous material (about 0.03 percent graphite and some fossiliferous resin) gives the ore much of its typical black coloration. Carbon is also present as carbonates.

Removing the organic material by any method, such as heating the pulp to 70°C. and skimming the "carbon" that floated, floating the "carbon" with flotation reagents, or by burning it off by heating, increased the percentage of soluble gold in the final product. It was exceedingly difficult, even when the ore had been treated with a series of solvents to remove various ingredients and then roasted, to lower the gold content below 0.02 or 0.03 ounce per ton as calculated back to weight of the original product. Perhaps other reagents may be found that will float more of the coated gold particles and interfering compounds so that a product still easier to cyanide will result.

Heating, even at the temperature of a hot plate, increased the gold dissolved by cyanide solution. Consideration of the results of many tests made on all scales from a 1-inch tube

furnace up to a full-size commercial kiln showed that the rate of dissolution increased up to a temperature of about 650°C. and then diminished. Employment of higher temperatures appeared to form slag around some of the particles. If ores are roasted at a low temperature they need not be crushed finer than 20-mesh, but if they are roasted above 650°C. increased gold extraction results from crushing the calcine to 65-mesh. Heating in atmospheres of moist chlorine gas or in steam also increased the amount of gold made soluble.

Some of the freshly mined Mercur ore contains 0.16 to 1.10 ounces of gold per ton and will allow some margin, but for the dump material containing only 0.08 ounce of gold the success or failure of the project will depend upon extracting every last possible thousandth of an ounce of gold. Further work on coated gold is contemplated; the installation of a blanket system in the flotation tailing launders of the Utah Copper Co. presents a good opportunity to study both the nature of the gold surfaces and to obtain material with which to work in the development of a suitable treatment process. Considerable oxidized iron distributed over the surfaces of the Utah Copper gold particles makes them resemble those from the Eureka Lily. Treating the gold particles with aqua regia leaves a similar siliceous-appearing residue. Gold particles are found coarser than 200-mesh, but these are not clean enough to float. Ninety percent of the feed is ground to pass 200-mesh; whence the chances to clean the particles by grinding does not appear good. The gold particles range from about 200- to what corresponds to 1,600-mesh.

Gold in Pyrite

In continuing the work on the occurrence of gold in pyrite[18] a method of identification was developed.

Gold was identified, even when occurring in very small amounts as in tailing samples, by the following procedure: The various samples were carefully classified as to particle size by elutriation into several products and the products panned with great care. The pyrite was then roasted and the iron present dissolved in mixtures of stannous chloride plus hydrochloric acid. The residues were examined by dark-field illumination and a daylight lamp. The gold particles thereupon appeared to be red and were easily identified from gangue. Tellurides of gold, if originally present, were indicated by the presence of sprouts on the gold particles; the occurrence was then checked in thin sections of the original ore. Grinding the original ore to a powder in an agate mortar followed by panning showed that particles, although not altered chemically, were deformed.

As in many samples of gold in pyrite, the gold in Wright-Hargreaves ore occurs along crystallographic planes in the pyrite. This ore is particularly interesting because gold is associated with part of the pyrite but not with all of it. When nearly all the gold freed by grinding is dissolved in cyanide solution and the cyanided tailing floated the pyrite in the flotation tailing contains much less gold per unit of pyrite than does the pyrite in the flotation concentrate. The gold in the flotation tailing is so finely disseminated in the pyrite that only a fifth of it is dissolved by cyanide solution in 100 hours from a sample ground to 20 microns. There was no evidence on examination in polarized light that the barren pyrite was in reality marcasite. Absence of "sprouting" on roasting indicated that no gold tellurides were present in the ore.

Lake Shore ore was found to be of the same general nature as the Wright-Hargreaves ore. There was no sign of "sprouting" on roasting, but tellurides of the base metals were present.

Tests have shown that the gold in Beattie ore can be cyanided profitably. The pyritic concentrate is shipped from the Atlantic seaboard to Tacoma, Wash. Studies of tho Beattie

---

18 Head, R. E., Form and Occurrence of Gold in Pyrite from a Metallurgical Standpoint: American Cyanimid Co., Ore Dressing Notes No. 2, September 1934.

ore revealed that even when the gold in pyrite is freed by grinding it is not necessarily soluble in cyanide solution. After the gold from such refractory particles had been dissolved by aqua regia it was found that some kind of siliceous film was left behind. Apparently this film had retarded the dissolution of the gold by cyanide solution. Similar coatings, although mostly of iron oxide, have been observed on gold particles from oxidized ores containing iron.

Several gold ores contain barrel pyrite. The plentiful pyrite in the Utah Apex will float readily and carry only a small percentage of gold. The pyrite in the Eureka Lily ore, when isolated under the microscope, separated, and analyzed, was also found to be nearly barren.

The sulphide concentrate from the Peavine district of Nevada presented somewhat of a problem. Three concentrates of successively lower grade containing some 92 percent of the gold could be made, but none of these products were satisfactory for shipping. It is interesting to note that although some of these ores contain many times as much silver as gold a higher percentage of the gold than of the silver usually can be floated.

Miscellaneous Ores

The increased price paid for gold has resulted in a marked increase in the number of small milling plants throughout the western States. Most of the ores treated are susceptible to treatment by usual methods, but the operator does not always select the best method. Numerous tests have been made on ores of this kind. A preliminary microscopic examination is followed by laboratory tests to determine a satisfactory treatment process.

The Searchlight (Nevada) ore contains only 0.07 ounce of gold per ton, but 65 percent of the material was floated in a concentrate containing 1.85 ounces per ton. By grinding to minus 100-mesh 77.4 percent of the gold was obtained in a concentrate containing 1.5 ounces of gold per ton. The ore contained soluble copper which consumed cyanide even though lead acetate was added. The old dumps at Tuscarora, Nev., are composed of material that has been roasted with and without salt. Over half of the gold and 12 percent of the silver were recovered by floating and passing the material over corduroy cloth. About 37 percent of the gold and 15 percent of the silver were soluble in cyanide solution. The tailings contained 5.85 ounces of silver and only 0.13 ounce of gold. The Candelaria tailings are largely of the same type. The material contained only 0.03 ounce of gold and 4.45 ounces of silver, yet 67 percent of the gold was cyanided and 40 percent floated, as against 26.8 and 19.1 percent, respectively, for the silver. The presence of oxidized copper in ores from the Ely (Nevada) district was found to be an important factor in the difficulty with which the silver minerals floated.

Amalgamation losses on Austin ore were severe and erratic. It was found that the recovery of gold could be increased to 90 percent by the use of gravity machines and that 96 percent of the gold was soluble in cyanide solution. The operator can choose the method that will best suit his means. Doten and Oversight ore from the Allegheny district of California are also unsuited for amalgamation. About 92 percent of their gold can be extracted by crushing to a size suited to percolation leaching with cyanide solution. Eastgate ore similarly will give up 97 percent of its gold by cyanidation after crushing to 30-mesh and 75 percent after crushing to 3/4-inch.

Earlier work by the Metallurgical Division showed that the Tallapoosa ore from Hog Mountain, Ala., could be cyanided directly, but that the consumption of reagents was high, and that 90 percent of the gold could be floated. The simplest effective means to overcome the high consumption of cyanide is to roast the concentrates at 650°C. and to wash with water before cyaniding; 96 percent of the gold was recovered by this procedure.

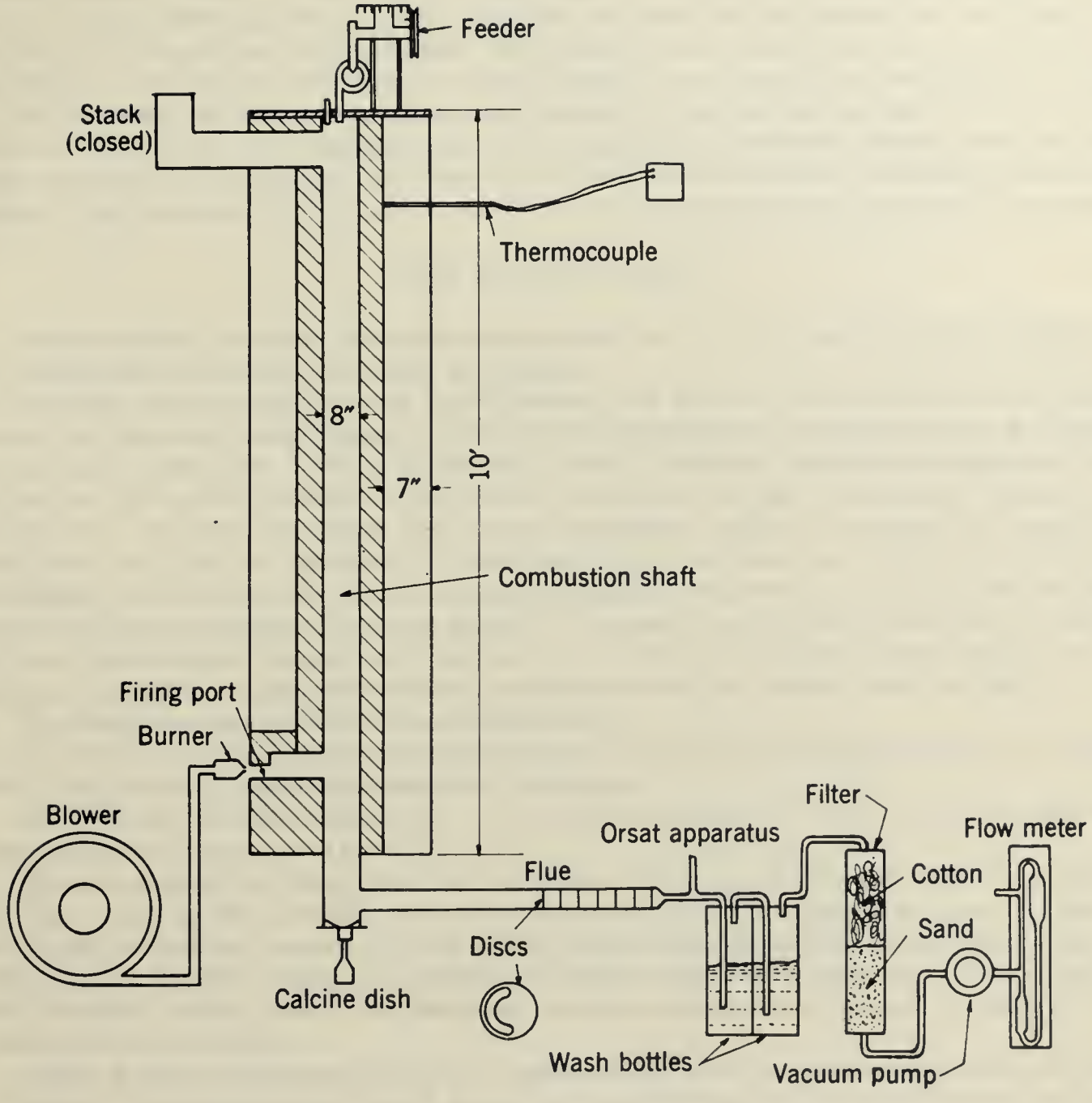


Figure 7.— Diagrammatic sketch of furnace and accessory apparatus.

The difficulty experienced with cyanide solutions was found to be due not to unusual impurities but to too little free lime and cyanide in solution. It was also found beneficial to apply earlier Bureau schemes for removing copper. The addition of lead acetate is beneficial for precipitating gold on zinc shavings, but when the solutions are well-evacuated and zinc dust is used as the precipitant lead acetate is of little service. As a result of experimental work at Reno on the Silver Peak (Nevada) ore a consultant was employed to work out the flow-sheet in detail. Amalgamation followed by cyanidation was found to be the most economical method for the treatment of Silver Peak ores. Cyaniding Crown Point ore gavo better results than flotation; the best extraction was obtained by leaching an all-slime product. The practical limitations of sand and slime leaching were also roughly determined for this ore.

COPPER METALLURGY SECTION

Flash roasting of copper flotation concentrates was the only problem studied duing the past 2 years by the section of copper metallurgy.

In flash roasting oxidation of the extremely fine sulphide particles which are characteristic of flotation concentrates is practically instantaneous, proceeding with a flash as each particle burns with a rapid evolution of heat. Previous reports have presented data showing the low sulphur content of the resulting calcine and the high sulphur dioxide concentration in the flue gas. This year's work, therefore, dealt with methods for separating copper and iron from the calcine in forms immediately suitable for further metallurgical processing. Obviously, since copper ores can be purified by flotation so as to contain practically nothing but copper, iron, and sulphur a successful method for extracting the copper from the flash-roaster calcine and leaving a pure iron oxide residue would constitute the basis of a possible new and advantageous process for reducing copper from its ores.

The attractive features of such a process are:

1. Removal of the undesirable impurities of the ore during concentration by flotation rather than by costly subsequent smelting operations.

2. Recovery in useful form of all the chemical elements of the concentrate, not only copper but also sulphur and iron.

The experiments performed this year indicate that this hope is realizable on a laboratory scale; the calcine, which consists chiefly of copper ferrites insoluble in leaching agents, may be rendered amenable to the ferric sulphate-sulphuric acid leaching process, developed by the Bureau of Mines, by passing the calcine mixed with coke dust through a heated shaft furnace a second time. The addition of small quantities of sodium chloride aids in giving a soluble product.

Figure 7 shows diagrammatically the flash-roasting shaft and accessories built for this year's work. The original shaft was 2 inches in internal diameter; the second model, 6 inches and this year's model, 8 inches. The flotation concentrate, suspended in air, is fed at the top into the shaft heated to 1,000°-1,150°C. and is oxidized during its descent to sulphur dioxide, which rises with the flue gases of which it constitutes 9 to 13 percent, and to small spherical shells of a copper ferrite mixture which are collected in the calcine-collecting box at the bottom of the shaft.

Table 4 shows the degree of sulphur elimination obtained in the 1934-35 flash-roasting shaft of figure 7.

TABLE 4.- Sulphur reduction in 8-inch shaft

Source and type of feed	Sulphur reduction, percent	Sulphur content of calcine, percent
Tacoma chalcopyrite	91.30	4.92
Miami pyrite	99.30	.90
Magma chalcopyrite, pyrite, and bornite	90.40	4.32
United Verde chalcopyrite	97.30	2.50

The figures of table 4 do not represent the degree of sulphur elimination attainable in flash-roasting; they indicate a defect in the design of the 8-inch shaft which was heated from below instead of from above; in the two smaller shafts 99 to 99.8 percent reduction of sulphur or 0.2 to 1 percent residual sulphur in the calcine was the rule.

Table 5 shows some results on reducing flash-roaster calcines by coke.

TABLE 5.- Reduction of calcine by coke

With sodium chloride added

Analysis of calcine feed, percent			Sulphuric acid - ferric sulphate leach, percent of total dissolved	
Copper	Iron	Sulphur	Copper	Iron
28.41	33.30	.....	88.2	5.4
32.38	32.52	0.40	72.9	0.0
32.50	28.50	.....	90.7	1.3
25.00	34.40	.....	84.7	1.6
33.51	35.40	.....	89.7	15.2
35.40	36.90	.....	92.0	16.8

Without sodium chloride

37.14	36.59	0.75	69.7	19.2
28.00	31.51	.....	90.7	31.7
30.92	34.21	.....	90.6	26.8
36.24	34.00	.....	89.3	34.5

These results are encouraging even though not satisfactory. Further study may be expected to give the conditions under which virtually all the copper can be extracted with probably a loss of no more than a third of the iron.

Further study of this promising process for treating copper flotation concentrates is being deferred until the two processes now under investigation for converting the residual iron oxide into sponge iron and the sulphur dioxide into elemental sulphur indicate the feasibility and desirability of building a pilot plant for the complete cycle - flotation concentrates converted into the three salable products, copper, sulphur, and iron.

The following papers published in May 1934 were released too late to be mentioned in the Director's report:

1. Dean, R. S., and Wartman, F. S. Outlook for Profitable Research in the Metallurgy of Copper. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 31-33.
2. Wartman, F. S., and Thompson, A. J. Preparation and Properties of Copper Ferrite. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 15-21.
3. Pryor, E. K., Sullivan, J. D., and Oldright, G. L. Leaching Copper Ores: Study of Oxidation of Iron Solutions Used as a Solvent. Rept. of Investigations, Bureau of Mines, May 1934, pp. 23-31.
4. Sloan, W. A., and Hallett, A. F. Regeneration of Ferric Sulphate in Copper-Leaching Solutions. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 33-35.
5. Brown, R. S., and Sullivan, J. D. Dissolution of Various Copper Minerals. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 37-51.
6. Sloan, W. A., and Davis, C. W. Hydrometallurgy of Copper Sulphide Ores and Its Relation to Mineral Structure. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 53-55.
7. Sloan, W. A. Comparison of Western Methods for the Determination of Oxidized Copper in Ores. Rept. of Investigations 3228, Bureau of Mines, May 1934, pp. 57-63.

LEAD AND ZINC SECTION

Although no experimental work was done by this section during the year the following papers reporting the results of previous work were published:

Doerner, H. A. Reduction of Zinc Ores by Natural Gas. Rept. of Investigations 3256, Bureau of Mines, December 1934, pp. 1-24.

Oldright, G. L., Brighton, T. B., and Dice, C. M. Recovery of Zinc from Ferrite Compounds in the Electrolytic Zinc Process. Rept. of Investigations 3256, Bureau of Mines, January 1935, pp. 26-43.

Oldright, G. L., Brighton, T. B., and Wilkes, D. P. Removal of Arsenic from Lead Blast-Furnace Charge. Eng. and Min. Jour., vol. 135, 1934, pp. 366-368.

Oldright, G. L., and Miller, Virgil. Progress Reports - Metallurgical Division. 7. Studies in Lead Metallurgy. Rept. of Investigations 3242, Bureau of Mines, October 1934, 11 pp.

_____Smelting in the Lead Blast Furnace - Handling Zinciferous Charges. 11. Preparation of the Charge by Sintering. Rept. of Investigations 3243, Bureau of Mines, December 1934, 60 pp.

_____Smelting in the Lead Blast Furnace - Handling Zinciferous Charges. 12. The Gases Within the Blast Furnace at Top of Tuyeres. Rept. of Investigations 3244, January 1935, 22 pp.

_____Smelting in the Lead Blast Furnace - Handling Zinciferous Charges. 13. Accretions at Various Elevations Within the Blast Furnace and Factors Governing Manner and Rate of Descent of Stock Column. Rept. of Investigations 3245, Bureau of Mines, January 1935, 15 pp.

_____Smelting in the Lead Blast Furnace - Handling Zinciferous Charges. 14. Methods of Charging in the Blast Furnace: Their Effect on Furnace Operation. Rept. of Investigations 3246, Bureau of Mines, January 1935, 12 pp.

_____Smelting in the Lead Blast Furnace - Handling Zinciferous Charges. 15. Slags from the Trail Blast Furnace. Rept. of Investigations 3264, Bureau of Mines, January 1935, 19 pp.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

322.09
Un32mo
Cop.2

Calvin Treison ✓

R. I. 3307 June 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

THE LIBRARY OF THE
AUG 12 1937
UNIVERSITY OF ILLINOIS

REPORT OF INVESTIGATIONS

VENTILATION OF MANHOLES

I. - EFFECT OF HOLES IN THE COVERS ON NATURAL VENTILATION

BY

G. W. JONES, W. E. MILLER, JOHN CAMPBELL,
AND W. P. YANT

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3307,
June 1936.

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

VENTILATION OF MANHOLES
I. - EFFECT OF HOLES IN THE COVERS ON NATURAL VENTILATION[1]

By G. W. Jones,[2] W. E. Miller,[3]
John Campbell,[4] and W. P. Yant[5]

INTRODUCTION

Because of its interest in accident prevention, the U.S. Bureau of Mines cooperates with city officials and utilities in investigating the causes of explosions in sewers, manholes, and other underground openings and in developing means for preventing such explosions. For several years this Bureau has been cooperating with the Edison Electric Illuminating Co. of Boston and the Boston Consolidated Gas Co. in detecting and eliminating combustibles and physiologically harmful gases in manholes in the city of Boston and adjacent territory.

These Bureau of Mines investigations[6] have shown that combustible gases from the usual sources - leakage from gas-distributing systems and from underground tanks and gasoline wastes - can be located and the dangerous conditions remedied by a well-planned, adequately supervised, systematic, and continuous manhole survey. However, other types of gas, which are termed here "soil" gases, also occur in manholes and are more difficult to eliminate because they have no fixed origin and because seasonal and barometric changes affect their production and passage into the manhole space. These gases appear to be produced by bacterial reactions and fermentation or through reactions of entrapped air in the soil. Soil gases are usually abnormally high in carbon dioxide and low in oxygen, and contain varying amounts of methane. They are especially dangerous to men entering manholes because they lack sufficient oxygen to support life. Owing to the difficulty of constructing gastight manholes, the most feasible method for eliminating soil gases from manholes is by ventilation.

Consequently the Bureau of Mines initiated a study of natural ventilation methods for the removal of these gases from manholes and also for taking care of unforeseen accumulations of combustible gases and vapors.

The following factors were considered by the Bureau to be important to the natural ventilation of manholes: (1) Relation of area of ventilation openings to the capacity of the manhole, (2) size and shape of the manhole, (3) design of the ventilation openings, (4) wind velocity, and (5) weather conditions.

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3307."

2 Chemist, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 Research assistant, cooperative employee.

4 Superintendent, Special Service Department, Edison Electric Illuminating Co. of Boston, Boston, Mass.

5 Supervising engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

6 Jones, G. W., and Perrott, G. St. J., Gases in Manholes: A Survey of a Utility in Boston, Mass. Rept. of Investigations 3109, May 1931, 16 pp.

Jones, G. W., Investigations During 1931 of Gases in Manholes in Boston, Mass.: Rept. of Investigations 3192, September 1932, 32 pp.

Jones, G. W., Campbell, John, and Goodwin, F. M., Investigations During 1932 of Combustibles in Manholes in Boston, Mass.: Rept. of Investigations 3213, June 1933, 17 pp. Investigations During 1933 of Combustibles in Manholes in Boston, Mass.: Rept. of Investigations 3260, October 1934, 25 pp.

EXPERIMENTAL MANHOLE

The experiments described in this report were carried out in an experimental manhole especially constructed for the purpose and situated back of the Pittsburgh Experiment Station in a ravine having rather steep hills to the north and south but open at the east and west. In many respects this location can be considered comparable to a manhole in a city street with rather tall buildings on both sides.

The manhole (fig. 1) was 8 feet deep and 3 feet square in cross section and had a capacity of 74.1 cubic feet. It was constructed of reinforced concrete with a standard iron manhole ring in the top slab. The cover was 27.5 inches in diameter and was supplied by the manufacturer with 6 round ventilation openings that had an average diameter of 0.812 inch. Forty-eight additional openings (0.75 inch in diameter) were drilled through the cover at equally spaced points, so that the effect of larger ventilation areas could be studied.

A 1/4-inch copper tube was used for feeding combustible gas into the manhole at a point near the center and 4 inches above the bottom. Three additional 1/8-inch copper tubes were installed for sampling the atmosphere at 1 foot below the cover, half way between the bottom and top, and 1 foot above the bottom of the manhole, respectively. All sample-tube openings were on the vertical center line of the manhole. Three thermocouples were installed for measuring the temperature of the atmosphere at the inlets to the gas-sampling lines. An additional thermocouple was placed 2-1/2 feet above the manhole cover for recording outside temperatures. A cup anemometer was installed on a cross bar 5 feet above the cover for measuring wind velocities. A wire cage was built around the manhole and anchored in concrete to prevent unauthorized persons from tampering with the instruments or receiving possible injury from the ignition of an explosive atmosphere. The thermocouple leads and gas line led from the manhole to a fireproof observation building, which housed apparatus for recording the manhole temperatures and for regulating the flow of gas into the manhole.

METHOD OF STUDYING MANHOLE VENTILATION

The method of studying manhole ventilation followed that used by the Bureau of Mines in a previous investigation[7] for determining the air changes in rooms contaminated by combustion products from space heaters. In Technical Paper 337 a formula was given for calculating the number of air changes in a room of known size when the amount of gas fed into the room at a constant rate and the percentages of gas present in the room at various intervals of time were known.

This formula, when applied to manholes, may be written as follows:

$$C = 100\ Q/R\ V\ (1 - e^{-Rt}),$$

where C = average percentage of combustible gas or vapor in the manhole after a given time, t,

R = air changes per day,
V = volume of manhole in cubic feet,
t = time in hours,
Q = cubic feet of combustible gas or vapor fed into manhole per day.

If gas is fed into a manhole at a known constant rate and in a manner that produces a homogeneous atmosphere, the concentration reaches a maximum when the amount of combustible

7 Jones, G. W., Berger, L. B., and Holbrook, W. F., Carbon Monoxide Hazards from House Heaters Burning Natural Gas: Tech. Paper 337, Bureau of Mines, 1923, p. 23.

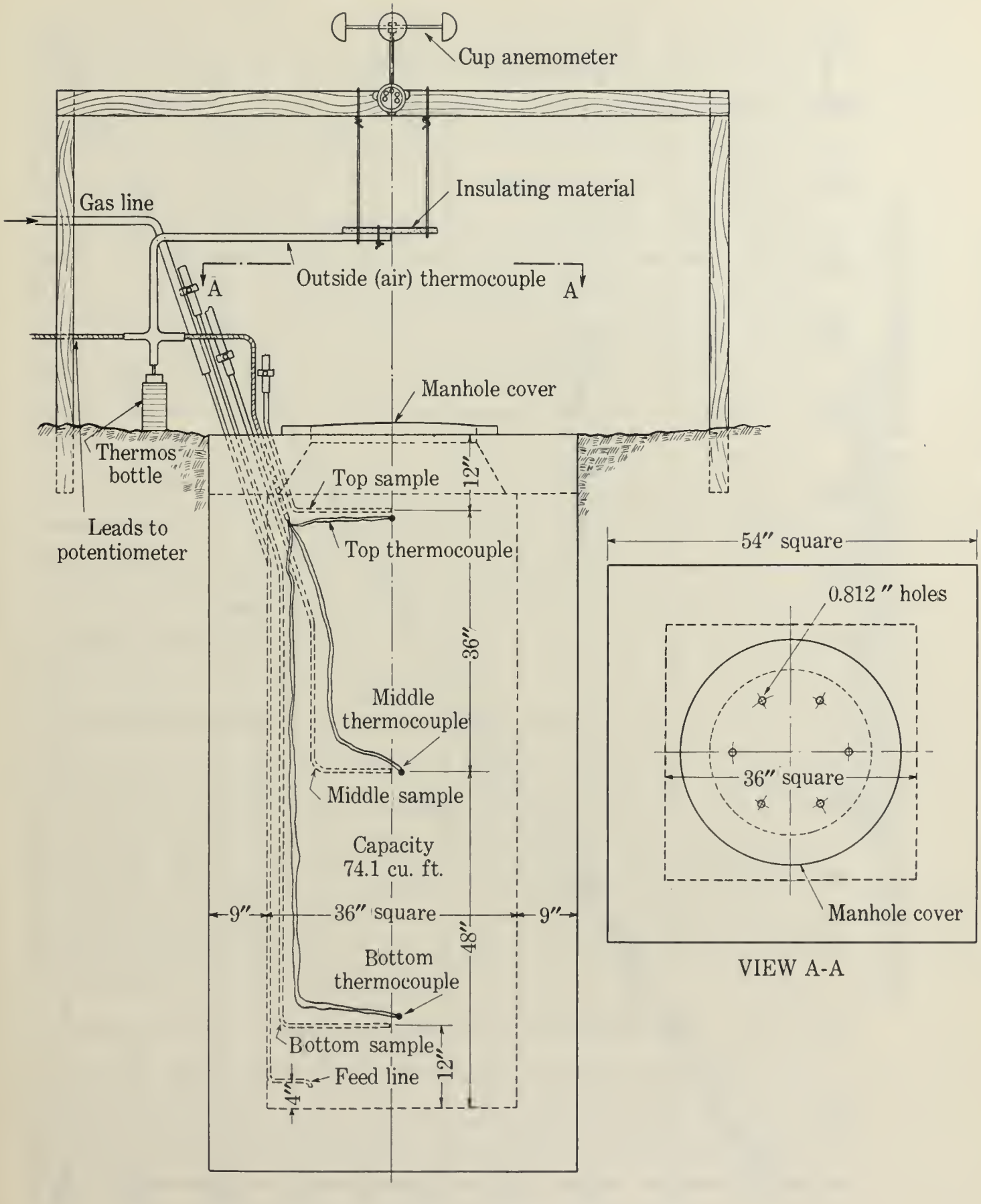


Figure 1.— Experimental manhole.

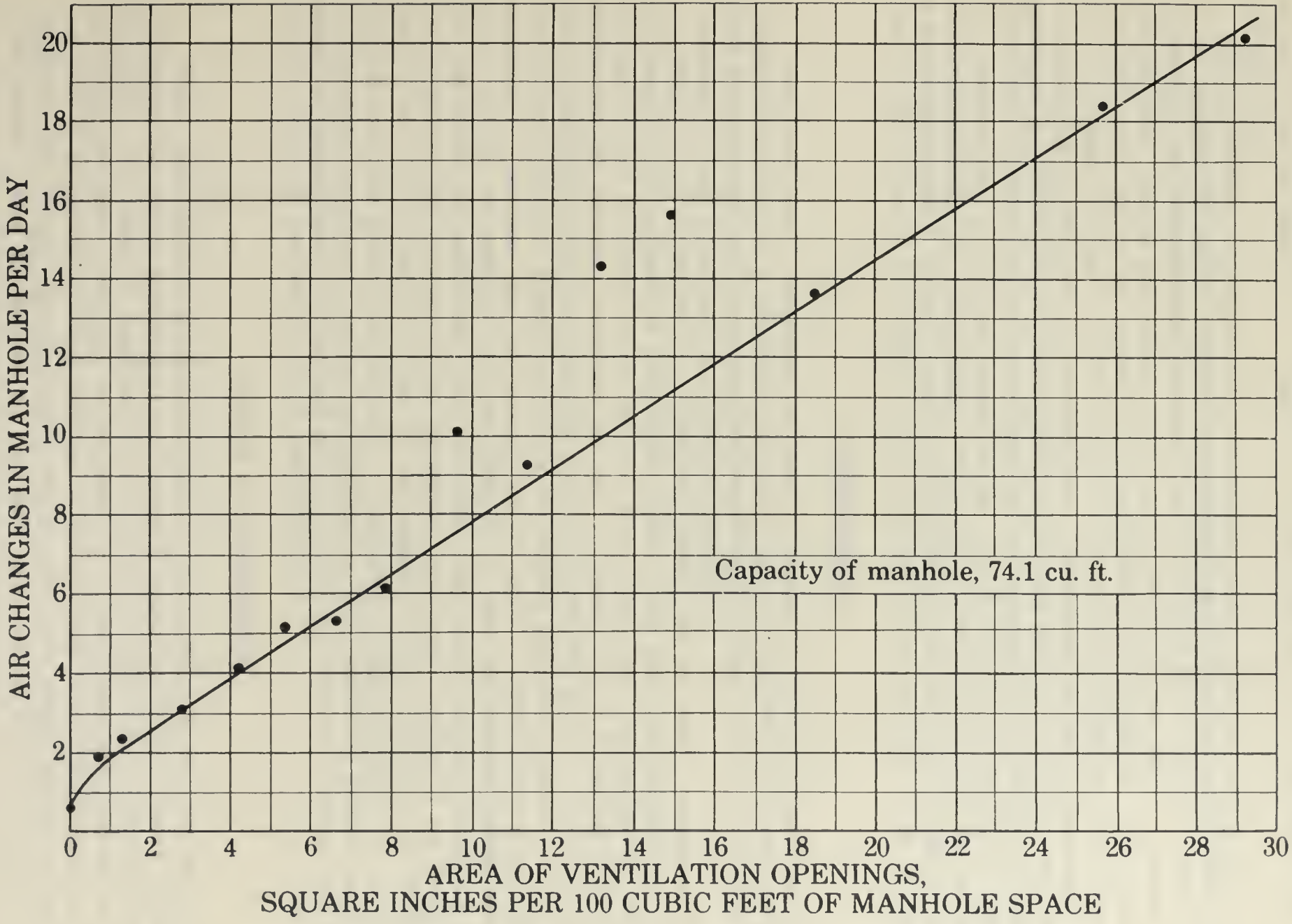


Figure 2.—Air changes in manhole per day versus area of ventilation openings.

gas escaping from the manhole equals the amount fed into the manhole. When this condition is reached, the last part of the formula, $(1- e^{-Rt})$, equals 1, and the formula becomes $R = 100 Q/C V$ and the number of air changes or the ventilation can be calculated from a determination of C.

As this formula can be used with accuracy only when the gas-air mixture is homogeneous, it was necessary to conduct a series of preliminary tests to determine the proper method of introducing the combustible gas to obtain a uniform mixture in the manhole. Tests were made with hydrogen, natural gas, ethane, and butane to determine the effect of density of the combustible gas used on the uniformity of distribution in the manhole. The results showed that when combustible gases lighter than air were fed into the manhole at the bottom under the conditions of these tests, a uniform mixture was present throughout the manhole space as determined by samples taken simultaneously at the top, middle, or bottom. The tests also showed that a combustible gas heavier than air (for example, butane, which has a density of 2.01 compared with 1.0 for air) should be fed into the manhole at the top in order to mix uniformly throughout the manhole space. Pittsburgh natural gas, having an average density of 0.64 (compared with 1 for air), was found suitable for test purposes if the gas was introduced at the bottom of the manhole; it was used for most of the tests given in this report.

Test Procedure

Tests were made to determine the effect of number of ventilation openings on the air changes that took place in the manhole per day. Each test lasted 2 to 7 days; the series of tests extended over more than a year and represented different climatic conditions, manhole temperatures, wind velocities, and barometric changes.

Before each test was begun the cover was removed and the manhole purged with fresh air for several hours by means of a mechanical blower. The cover was replaced, and all cover openings, except the number chosen for test, were closed with rubber stoppers. The gas flow was started and adjusted to the desired rate, as indicated by a calibrated flowmeter. The actual quantities of gas were measured by a calibrated wet meter. In the course of each test the percentage of combustible gas in the manhole atmosphere started at zero and then increased to a maximum, when the leakage from the manhole, due to ventilation, equaled the amount fed into the manhole. The time required to reach this maximum concentration varied with the amount of ventilation. For all tests except those with no openings in the cover the maximum average concentration was reached within 4 hours. Samples of the atmosphere were taken at the top, middle, and bottom of the manhole 4 and 8 hours after the initiation of the test and three times a day thereafter. Coincident with the taking of the samples, measurements and records were made of the temperature, both inside the manhole and 2 1/2 feet above the manhole cover, the gas rate per hour, barometric pressure, average wind velocity, relative humidity, weather conditions, and rainfall.

RESULTS OF INVESTIGATIONS

Table 1 and figure 2, giving the results of tests made, show that the ventilation (air changes per day) in the manhole is almost directly proportional to the number of openings in the cover. The number of openings in the cover is the factor that chiefly affected the ventilation and shows strikingly the advisability of having ventilated manhole covers. In test 6-S, in which there were no openings (there was some ventilation around the edges of the cover), the average concentration of natural gas in the manhole during the test equaled 31.20 percent. When one opening was provided (tests 1-S and 2-S), the concentration was 10.11 percent; when eight openings were provided, the concentration was reduced to 3.77 percent; 24 holes, 1.25 percent; and 48 holes, 0.98 percent. The provision of 48 holes, as shown in the last column of table 1, increased the ventilation over 30 times that which was present when no holes were provided.

TABLE 1.- Ventilation tests showing variation in the concentration of combustible gas in the top, middle, and bottom of the manhole and air changes per day, with change in the number of ventilation openings. Capacity of manhole, 74.1 cubic feet; regular typo cover used

Test number	Number of ventilation holes in cover	Area of openings, square inches	Area of openings, square inches per 100 cubic feet of manhole space[1]	Combustible gas used in test	Density of gas used, air = 1.0	Average amount of combustible gas fed into manhole per day, cubic feet	Average maximum concentration of combustible gas in manholes, percent by volume[2]				Duration of test, days	Average wind velocity, miles per hour	Ventilation, air changes in manhole per day
							Top	Middle	Bottom	Average			
9-S	48[3]	21.66	29.25	Natural gas	0.64	14.60	0.86	1.04	1.05	0.98	7.00	2.5	20.10
25-S	42	19.01	25.65	do.	.64	14.47	1.06	1.08	1.05	1.06	3.33	4.1	18.40
22-S	30	13.72	18.50	do.	.64	14.50	1.42	1.45	1.40	1.42	4.00	1.5	13.77
8-S	24	11.07	14.95	do.	.64	14.40	1.29	1.12	1.35	1.25	7.00	2.6	15.55
26-S	21	9.74	13.14	do.	.64	14.40	1.36	1.38	1.34	1.36	5.00	1.8	14.30
14-S	18	8.41	11.35	do.	.64	14.47	2.04	2.17	2.12	2.11	5.00	1.6	9.25
24-S	15	7.09	9.57	do.	.64	14.35	1.90	1.88	1.97	1.92	3.33	.5	10.08
7-S	12	5.76	7.77	do.	.64	14.70	3.26	3.26	3.21	3.24	6.33	1.6	6.11
13-S	10	4.88	6.60	do.	.64	14.60	3.55	3.75	3.60	3.63	4.33	1.6	5.42
12-S	8	3.99	5.39	do.	.64	14.50	3.69	3.83	3.79	3.77	5.00	2.2	5.19
18-S	6	3.11	4.20	Hydrogen	.07	14.40	4.24	4.34	4.11	4.23	2.00	.9	4.60
5,16,28; 29,30,31, 32,34,35	6	3.11	4.20	Natural gas	.64	16.06	4.99	5.48	5.25	5.24	3.89	1.8	4.14
19-S	6	3.11	4.20	Ethane	1.05	14.40	5.29	5.78	7.20	6.07	4.33	1.9	3.20
20-S	6	3.11	4.20	Butane	2.01	14.02	4.23	4.80	10.26	6.43	4.33	2.0	2.94
4-S	4	2.07	2.80	Natural gas	.64	15.10	6.42	7.00	6.55	6.66	5.00	2.4	3.06
3-S	2	1.04	1.40	do.	.64	14.70	8.45	8.62	8.04	8.37	5.00	2.0	2.37
1-S, 2-S	1	.52	.70	do.	.64	14.40	10.18	10.22	9.93	10.11	5.00	1.9	1.92
6-S	0	4/	4/	do.	.64	14.60	31.29	31.27	30.93	31.20	3.00	2.0	.63

1 Measured at prevailing temperatures and pressures.

2 Analyses on a dry basis.

3 6 holes 0.812 inch in diameter; remaining holes 0.75 inch.

4 Small amount of air leakage around rim of cover.

This is one of a series of reports dealing with methods of obtaining better ventilation in manholes by natural means. Others will follow dealing with the effect of size of the manhole on the ventilation rate, and the effect of wind velocity, specially designed manhole cover openings, and ventilation ducts.

SUMMARY

Tests were made on a manhole (capacity, 74.1 cubic feet) to determine what change in the natural ventilation takes place due to varying the number of ventilation openings in the manhole cover.

The results showed that the ventilation (air changes per day) in the manhole was almost directly proportional to the number of openings in the cover, and also that the number of openings in the cover was the factor that chiefly affected the ventilation.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

622.09
Un32rno
Cop2

R. I. 3308

JULY 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

THE NATIONAL SAFETY COMPETITION OF 1935

BY

W. W. ADAMS, T. D. LAWRENCE, AND D. R. WHITE

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R. I. 3308.
July 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

THE NATIONAL SAFETY COMPETITION OF 1935[1]

By W. W. Adams,[2] T. D. Lawrence,[3] and D. R. White[3]

This paper presents the results of the eleventh yearly accident-prevention contest, the National Safety Competition of 1935, conducted by the United States Bureau of Mines. Participating in the competition were 336 mines and Quarries in 37 States. Five mining or quarrying companies, each the leader in its group for having established the best safety record during the year, were awarded the national safety trophies 'Sentinels of Safety'.

Since these annual contests were begun 11 years ago the scope of the competition has been broadened materially, both as to the number of plants participating and as to the number of man-hours of exposure to accident hazards. Moreover, marked progress in the prevention of accidents has been achieved, as is indicated by an almost constant downward course in the accident-frequency rates for the contest group as a whole during the past 11 years. On the other hand, the accident-severity rates for the group showed an upward trend during the first part of the period and a downward trend during the latter part, except in 1930 and 1934, when two disastrous explosions occurred with heavy loss of life, and in 1935, when there was an increase in the number of fatalities although no major disasters occurred. The accident rate for 1935, both as to frequency and severity, was higher than for 1934.

The National Safety Competition is conducted by the Bureau of Mines in the interest of safety in the mining and quarrying industries. This object is accomplished directly through the efforts of the competing companies and their employees to win first place in the competition and thereupon receive well-merited and national recognition for their achievement. Indirectly the purpose of the contest is accomplished through studies by the Bureau of Mines of the facts contained in the accident reports which the competing companies furnish to enable the Bureau to compute the accident rates that are the basis for determining the winners of the contest.

The statistics that constitute the basis for determining the winning companies and the relative standing of all companies are prepared uniformly without regard to the States in which the companies operate, or to differences in classes of accidents covered by State laws relating to compensation for injuries resulting from industrial accidents.

Bronze trophies, known as 'Sentinels of Safety', were provided by the Explosives Engineer magazine to be awarded annually by the Bureau of Mines. These trophies were presented to five companies in the contest of 1935, each company having established the best accident-prevention record within one of the five classes into which all companies were divided, namely, anthracite-mining companies, bituminous-coal-mining companies, metal-mining companies, operators of nonmetallic-mineral mines other than coal mines, and quarries or open-

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3308".

2 Supervising statistician, employment statistics section, U.S. Bureau of Mines.

3 Employment statistics section, U.S. Bureau of Mines.

cut mines. The trophies are awarded for one year and are then transferred to the winners of the succeeding year's contest.

The trophy for bituminous-coal mines was won by a company in West Virginia. A company in Minnesota won the trophy for metal mining. In the nonmetal-mine group the trophy was won by a company in New York. A quarry in California won the trophy for the quarry-and-opencut-mine group. The trophy for anthracite mines remains in Pennsylvania, as all anthracite mines in the contest are in that State.

In determining the relative standing of the various companies enrolled in the competition, all mines and quarries were graded according to their accident-severity rates, that is, according to the number of days of disability of employees on account of accidents in proportion to the number of man-hours of work done by all employees at the plant. Reports of accidents and of man-hours worked were furnished to the Bureau of Mines by the companies. These reports were examined and classified carefully to insure uniform weighting of accidents according to the number of days of disability to be charged and to insure uniformity of methods of calculating the accident-severity rates for the various mines and quarries. When the relative standing of all plants had been determined in this way a summary of the records of the companies having the lowest accident-severity rates in their respective groups was sent to the members of a committee of award for examination in accordance with the rules governing the safety contest. The members of this committee for 1935 were W. H. Cameron, managing director of the National Safety Council; William Green, president of the American Federation of Labor; Julian D. Conover, secretary of the American Mining Congress; T. T. Read, Vinton professor of mining engineering of Columbia University; J. D. Battle, executive secretary of the National Coal Association; A. J. R. Curtis, assistant to the general manager of the Portland Cement Association; and A. T. Goldbeck, director of the bureau of engineering of the National Crushed Stone Association, Inc. When two or more mines within the same group operated without an accident during the year the contest rules provided that the trophy for the group should be awarded to the plant whose record showed the largest number of man-hours worked.

WINNERS OF TROPHIES

The trophy for bituminous-coal mines was awarded to the Ingram Branch mine, Ingram Branch, Fayette County, W. Va. This mine had no disabling accidents in 1935 and worked 235,211 man-hours, the period of operation being 231 days. The mine was operated during the year by the Elkhorn Piney Mining Coal Co.

The trophy for anthracite mines was awarded to the Jeddo No. 7 mine, Harleigh, Luzerne County, Pa. This mine was operated by the Jeddo-Highland Coal Co. and worked 184,992 man-hours in 1935, with 6 lost-time accidents causing 66 days of disibility to the employees. The accident rate on which the trophy was awarded was 0.357 per thousand man-hours of exposure of the employees to risk, the period of operation being 164 days.

For metal mining, the trophy was won by the Zenith mine, Ely, St. Louis County, Minn. No disabling accidents occurred at this mine during 1935. The mine was operated by the Vermillion Mining Co. and was in operation for 162 days and worked 196,201 man-hours.

The trophy for mining nonmetallic minerals was won by the Akron Gypsum mine, Akron, Erie County, N. Y. This mine worked 79,144 man-hours in 1935 without a disabling accident among its employees. It was operated by the Certain-teed Products Corporation and was in operation 246 days.

For quarring and opencut mining, the trophy for 1935 was won by the Folsom State Prison quarry, Folsom, Sacramento County, Calif. The operating time for this quarry was 717,168 man-hours, or 259 days, and no disabling accident occurred during the year. The quarry is operated by the Folsom State Prison.

WINNERS OF HONORABLE MENTION

Companies receiving honorable mention were:

Anthracite Mines

Stillwater mine, Wayne County, Pa., near Vandling, Lackawanna County, Pa., operated by the Hudson Coal Co. Mine operated 314,447 man-hours in 1935 with 22 lost-time accidents causing 216 days of disability. The accident severity rate was 0.687.

Miles Slope mine, Dickson, Lackawanna Couhty, Pa., operated by the Hudson Coal Co. Mine operated 502,837 man-hours in 1935 with 19 lost-time accidents causing 401 days of disability. The accident severity rate was 0.797.

Highland No. 6 mine, Freeland, Luzerne County, Pa., operated by the Jeddo-Highland Coal Co. Mine operated 103,024 man-hours in 1935 with 5 lost-time accidents causing 84 days of disability. The accident severity rate was 0.815.

Clinton mine, Vandling, Lackawanna County, Pa., operated by the Hudaon Coal Co. Mine operated 318,137 man-hours in 1935 with 18 lost-time accidents causing 315 days of disability. The accident-severity rate was 0.990.

Bituminous-Coal mines

Thacker mine, Pike County, Ky., near Williamson, Mingo County, W. Va., operated by the Pond Creek Colliery of the Norfolk and Western Railway Co. Mine operated 225,631 man-hours in 1935 without a lost-time accident.

No. 5 mine, Pomeroy, Meigs County, Ohio, operated by the Lost Run Coal Co. Mine operated 180,964 man-hours in 1935 with 4 lost-time accidents causing 72 days of disability. The accident-severity rate was 0.398.

Vulcan mine, Pike County, Ky., near Vulcan, Mingo County, W. Va., operated by the Vulcan Colliery of the Norfolk and Western Ry. Co. Mine operated 157,052 man-hours in 1935 with 3 lost-time accidents causing 93 days of disability. The accident severity rate was 0.592.

"C" mine, Superior, Sweetwater County, Wyo., operated by the Union Pacific Coal Co. Mine operated 280,684 man-hours in 1935 with 3 lost-time accidents causing 197 days of disability. The accident-severity rate was 0.702.

Metal Mines

Fontana mine, Fontana, Swain County, N. C., operated by the North Carolina Exploration Co. Mine operated 123,152 man-hours in 1935 without a lost-time accident.

Mississippi mine, Keewatin, Itasca County, Minn., operated by the M. A. Hanna Co. Mine operated 75,536 man-hours in 1935 without a lost-time accident.

Sunday Lake mine, Wakefield, Gogebic County, Mich., operated by the Sunday Lake Iron Co. Mine operated 230,580 man-hours in 1935 with 2 lost-time accidents causing 43 days of disability. The accident-severity rate was 0.186.

Ironton & Tilden mine, Bessemer, Gogebic County, Mich., operated by the Republic Steel Corporation. Mine operated 372,803 man-hours in 1935 with 2 lost-time accidents causing 77 days of disability. The accident-severity rate was 0.207.

Nonmetallic-Mineral Mines[4]

Wampum mine, Wampum, Lawrence County, Pa., operated by the Crescent Portland Cement Co. Mine operated 87,271 man-hours in 1935.

Ironton mine, Ironton, Lawrence County, Ohio, operated by the Alpha Portland Cement Co. Mine operated 47,575 man-hours in 1935.

Akron mine, Akron, Erie County, N. Y., operated by the Universal Gypsum & Lime Co. Mine operated 46,632 man-hours in 1935.

Sugar Creek mine, Independence, Jackson County, Mo., operated by the Missouri Portland Cement Co. Mine operated 46,128 man-hours in 1935.

Lower Gypsum mine, Gypsum, Ottawa County, Ohio, operated by the United States Gypsum Co. Mine operated 37,852 man-hours in 1935.

Chattanooga mine, Chattanooga, Hamilton County, Tenn., operated by the Signal Mountain Portland Cement Co. Mine operated 37,699 man-hours in 1935.

Quarries and Opencut Mines[5]

Mesabi Chief iron-ore mine, Keewatin, Itasca County, Minn., operated by the Hanna Ore Mining Co. Mine operated 258,894 man-hours in 1935.

Mahoning iron-ore mine, Hibbing, St. Louis County, Minn., operated by the Mahoning Ore and Steel Co. Mine operated 198,398 man-hours in 1935.

Bethlehem limestone quarry, Bethlehem, Northampton County, Pa., operated by the Bethlehem Steel Co. Quarry operated 175,086 man-hours in 1935.

Krause No. 1 limestone quarry, Columbia, St. Clair County, Ill., operated by the Columbia Quarry Co. Quarry operated 167,671 man-hours in 1935.

Sagamore iron-ore mine, Ironton, Crow Wing County, Minn., operated by the Sagamore Ore Mining Co. Mine operated 129,336 man-hours in 1935.

Holston limestone quarry, Mascot, Knox County, Tenn., operated by the American Limestone Co. Quarry operated 122,934 man-hours in 1935.

Naginey limestone quarry, Mifflen County, Pa., operated by the Bethlehem Steel Co. Quarry operated 120,733 man-hours in 1935.

Scranton iron-ore mine, Hibbing, St. Louis, Minn., operated by the Hoyt Mining Co. Mine operated 118,747 man-hours in 1935.

Albany iron-ore mine, Hibbing, St. Louis County Minn., operated by the Crete Mining Co. Mine operated 114,732 man-hours in 1935.

Susquehanna iron-ore mine, Hibbing, St. Louis County, Minn., operated by the Republic Steel Corporation. Mine operated 114,632 man-hours in 1935.

Cape Girardeau limestone quarry, Cape Girardeau, Cape Girardeau County, Mo., operated by the Marquette Cement Manufacturing Co. Quarry operated 111,928 man-hours in 1935.

Oglesby limestone quarry, Oglesby, LaSalle County, Ill., operated by the Marquette Cement Manufacturing Co. Mine operated 107,263 man-hours in 1935.

Reformatory limestone quarry, Anamosa, Jones County, Iowa, operated by the Iowa State Reformatory. Quarry operated 105,598 man-hours in 1935.

Bridgeport limestone quarry, Bridgeport, Montgomery County, Pa., operated by the Bethlehem Steel Co. Quarry operated 98,953 man-hours in 1935.

North Branford trap-rock quarry No. 7, North Branford, New Haven County, Conn., operated by the New Haven Trap Rock Co. Quarry operated 95,061 man-hours in 1935.

---

4 All operated with no lost-time accidents.

5 All operated with no lost-time accidents.

Earlham limestone quarry, Earlham, Madison County, Iowa, operated by the Hawkeye Portland Cement Co. Quarry operated 91,399 man-hours in 1935.

Mahnomen iron-ore mine, Ironton, Crow Wing County, Minn., operated by the Cuyuna Ore Co. Mine operated 87,546 man-hours in 1935.

Piqua limestone quarry, Piqua, Miami County, Ohio, operated by the Ohio Marble Co. Quarry operated 87,370 man-hours in 1935.

Ruegg limestone quarry, Ruegg, St.Louis County, Mo., operated by the Missouri Portland Cement Co. Quarry operated 86,386 man-hours in 1935.

Buckeye Creek limestone quarry, East Fultonham, Muskingum County, Ohio, operated by the Pittsburgh Plate Glass Co. Quarry operated 84,869 man-hours in 1935.

Berkeley Nos. 5 & 6 limestone quarry, Martinsburg, Berkeley County, W. Va., operated by the North American Cement Corporation. Quarry operated 84,396 man-hours in 1935.

Monocacy traprock quarry, Birdsboro, Berks County, Pa., operated by the John T. Dyer Quarry Co. Quarry operated 79,140 man-hours in 1935.

Security limestone quarry, Security, Washington County, Md., operated by the North American Cement Corporation. Quarry operated 75,967 man-hours in 1935.

Blue Mount serpentine quarry, White Hall, Baltimore County, Md., operated by the J. E. Baker Co. Quarry operated 69,811 man-hours in 1935.

Fogelsville cement-rock quarry, Fogelsville, Lehigh County, Pa., operated by the Lehigh Portland Cement Co. Quarry operated 65,268 man-hours in 1935.

Cowell limestone quarry, Cowell, Contra Costa County Calif., operated by the Cowell Portland Cement Co. Quarry operated 62,899 man-hours in 1935.

Pembroke limestone quarry, Pembroke, Giles County, Va., operated by the Pembroke Limestone Corporation. Quarry operated 57,982 man-hours in 1935.

White Haven limestone quarry, White Haven, Luzerne County, Pa., operated by the General Crushed Stone Co. Quarry operated 57,012 man-hours in 1935.

Plymouth iron-ore mine, Wakefield, Gogebic County, Mich., operated by the Plymouth Mining Co. Mine operated 53,835 man-hours in 1935.

Union limestone quarry, Mt. Wolf, York County, Pa., operated by the J. E. Baker Co. Quarry operated 53,487 man-hours in 1935.

Greencastle limestone quarry, Greencastle, Putnam County, Ind., operated by the Mid-West Products Corporation. Quarry operated 53,201 man-hours in 1935.

Middlefield No. 1 trap-rock quarry, New Haven County, Conn., near Middlefield, Middlesex County, Conn., operated by the New Haven Trap Rock Co. Quarry operated 52,575 man-hours in 1935.

No. 4 limestone quarry, Nazareth, Northampton County, Pa., operated by the Pennsylvania-Dixie Cement Corporation. Quarry operated 51,808 man-hours in 1935.

Clinchfield limestone quarry, Clinchfield, Houston County, Ga., operated by the Pennsylvania-Dixie Cement Corporation. Quarry operated 51,403 man-hours in 1935.

Nazareth limestone quarry, Nazareth, Northampton Couhty, Pa., operated by the Lone Star Cement Corporation. Quarry operated 50,299 man-hours in 1935.

Union Bridge limestone quarry, Frederick County, Md., near Union Bridge, Carroll County, Md., operated by the Lehigh Portland Cement Co. Quarry operated 49,270 man-hours in 1935.

Bennett iron-ore mine, Keewatin, Itasca County, Minn., operated by the Bennett Mining Co. Mine operated 47,421 man-hours in 1935.

Union Furnace limestone quarry, Huntingdon County, Pa., near Tyrone, Blair County, Pa., operated by the American Lime & Stone Co. Quarry operated 47,278 man-hours in 1935.

Petoskey limestone quarry, Petoskey, Emmet County, Mich., operated by the Petoskey Portland Cement Co. Quarry operated 46,953 man-hours in 1935.

Birmingham limestone quarry, Birmingham, Jefferson County, Ala., operated by the Lehigh Portland Cement Co. Quarry operated 46,790 man-hours in 1935.

Glens Falls limestone quarry, Saratoga County, N. Y., near Glens Falls, Warren County, N. Y., operated by the Glens Falls Portland Cement Co. Quarry operated 46,626 man-hours in 1935.

Thomasville limestone quarry, Thomasville, York County, Pa., operated by the J. E. Baker Co. Quarry operated 46,626 man-hours in 1935.

Plainville No. 4 trap-rock quarry, Plainville, Hartford County, Conn., operated by the New Haven Trap Rock Co. Quarry operated 45,079 man-hours in 1935.

Inwood limestone quarry, Inwood, Berkeley County, W. Va., operated by the J. E. Baker Co. Quarry operated 45,054 man-hours in 1935.

Lone Star limestone quarry, Hudson, Columbia County, N. Y., operated by the Lone Star Cement Co. of N. Y., Inc. Quarry operated 44,297 man-hours in 1935.

Martins Creek No. 4 cement-rock quarry, Martins Creek, Northampton County, Pa., operated by the Alpha Portland Cement Co. Quarry operated 42,864 man-hours in 1935.

Gasport limestone quarry, Gasport, Niagara County, N. Y., operated by the Wickwire Spencer Steel Co. Quarry operated 42,491 man-hours in 1935.

San Vicente limestone quarry, Davenport, Santa Cruz County, Calif., operated by the Santa Cruz Portland Cement Co. Quarry operated 39,984 man-hours in 1935.

Dixon limestone quarry, Dixon, Lee County, Ill., operated by the Medusa Portland Cement Co. Quarry operated 39,841 man-hours in 1935.

Catskill limestone quarry, Alsen, Greene County, N. Y., operated by the North American Cement Corporation. Quarry operated 39,277 man-hours in 1935.

Greencastle limestone quarry, Greencastle, Putnam County, Ind., operated by the Lone Star Cement Company Indiana, Inc. Quarry operated 38,079 man-hours in 1935.

Mitchell limestone quarry (Lehigh Lime Co.), Mitchell, Lawrence County, Ind., operated by the Lehigh Portland Cement Co. Quarry operated 35,973 man-hours in 1935.

La Salle limestone quarry, La Salle, La Salle County, Ill., operated by the Alpha Portland Cement Co. Quarry operated 35,685 man-hours in 1935.

Corsica iron-ore mine, Elcor, St. Louis County, Minn., operated by the Corsica Iron Co. Mine operated 35,097 man-hours in 1935.

Steelton limestone quarry, Steelton, Dauphin County, Pa., operated by the Bethlehem Steel Co. Quarry operated 34,974 man-hours in 1935.

Marcem limestone quarry, Gate City, Scott County, Va., operated by the Pennsylvania-Dixie Cement Corporation. Quarry operated 33,558 man-hours in 1935.

New Castle cement-rock quarry, New Castle, Lawrence County, Pa., operated by the Lehigh Portland Cement Co. Quarry operated 32,130 man-hours in 1935.

Birmingham limestone quarry, Birmingham, Jefferson County, Ala., operated by the Lone Star Cement Corporation. Quarry operated 30,531 man-hours in 1935.

Medusa limestone quarry, York, York County, Pa., operated by the Medusa Portland Cement Co. Quarry operated 30,478 man-hours in 1935.

Cowan limestone quarry, Cowan, Franklin County, Tenn., operated by the Cumberland Portland Cement Co. Quarry operated 30,129 man-hours in 1935.

Richmond iron-ore mine, Palmer, Marquette County, Mich., operated by the Richmond Iron Co. Mine operated 30,077 man-hours in 1935.

SCOPE OF CONTEST

The wide scope of the National Safety Competition of 1935 is indicated by the fact that 37 States were represented by the 336 mines and quarries that participated in the contest. The 37 States were:

Alabama	Illinois	Massachusetts	North Dakota	Virginia
Arkansas	Indiana	Michigan	Ohio	Washington
California	Iowa	Minnesota	Oklahoma	West Virginia
Colorado	Kansas	Missouri	Pennsylvania	Wisconsin
Connecticut	Kentucky	Montana	Tennessee	Wyoming
Florida	Louisiana	New Mexico	Texas	
Georgia	Maine	New York	Utah	
Idaho	Maryland	North Carolina	Vermont	

Salient features of the 1935 competition are given in Table 1. The figures show an increase of more than 10 million man-hours represented in the contest of 1935 compared with the number of man-hours represented in the contest of 1934.

TABLE 1.- Accident data compared by types of mines

Type of mine	Number of mines	Man-hours	Fatal injuries	Non-fatal injuries	Total	Days of disability	Frequency rate	Severity rate
Anthracite	37	24,475,637	34	1,723	1,757	289,491	71.786	11.828
Bituminous coal	109	42,342,213	55	2,162	2,217	505,834	52.359	11.946
Metal	24	8,999,184	8	240	248	67,126	27.558	7.459
Nonmetallic mineral	20	1,984,540	0	23	23	1,145	11.590	.577
Total	190	77,801,574	97	4,148	4,245	863,596	54.562	11.100
Quarries and opencut	146	14,177,300	4	196	200	51,764	14.107	3.651
Grand total 1935	336	91,978,874	101	4,344	4,445	915,360	48,326	9.952
1934	[1]333	81,759,857	66	3,668	3,734	628,905	45.670	7.692
1933	332	78,826,713	75	4,569	4,644	701,593	58.914	8.900
1932	322	68,081,721	66	3,763	3,829	585,481	56.241	8.600
1931	350	79,544,551	76	3,871	3,947	692,092	49.620	8.701
1930	[2]337	96,533,706	102	5,869	5,971	849,778	61.854	8.803
1929	311	106,533,006	114	6,954	7,068	986,306	66.346	9.258
1928	284	91,746,707	110	5,999	6,109	900,182	66.585	9.812
1927	256	91,919,812	104	6,541	6,645	8,6,021	72,291	9,421
1926	256	95,055,815	93	7,220	7,313	799,493	76.934	8.411
1925	210	68,418,283	68	6,780	6,848	580,895	100.090	8.490

1 Omitting the entire record covering one mine in which 17 men were killed in a single disaster.

2 Omitting the record of one mine in which 73 lives were lost in a single explosion.

COMPARISON OF ACCIDENT-SEVERITY RATES

A comparison of the accident-severity rates of individual plants with the average severity rate for their respective groups as a whole indicates that 63 percent of the plants within each group had more favorable severity rates than the average rates for the group. In the anthracite group, for example, which included 37 mines for which the group severity-rate was 11.828, there were 25 mines whose rates were better than the group average and 12 whose rates were worse than the average. Likewise, in the bituminous group, which included 109 mines, 70 mines had better and 39 worse rates than the average for the group. Among 24 metal mines 18 had better-than-average rates and 6 had worse-than-average rates compared with the group rate, which was 7.459 per thousand man-hours of exposure. Out of 20 nonmetal mines the rates for 13 were better than the average and 7 were worse than the average. In the group covering 146 quarries and opencut mines 129 had better-than-average rates and 17 plants had individual rates that were less favorable than the average rate for the group.

Tables 2 to 6, which follow, show the individual record of each mine or quarry that participated in the National Safety Competition of 1935 and the relative position each plant occupied when arranged according to the accident-severity rates of the various plants.

Comparative accident rates for the 11 years during which the contest has been conducted are given in tables 7 and 8.

Table 2.- Accident data for certain ANTHRACITE MINES in 1935.[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
1	36,467				1	1				4	4	27.422	0.110
2	184,992				6	6				66	66	32.434	0.357
3	314,447				22	22				216	216	69.964	0.687
4	502,837				19	19				401	401	37.786	0.797
5	103,024				5	5				84	84	48.532	0.815
6	318,137				18	18				315	315	56.579	0.990
7	3,025,405			3	160	163			900	2,610	3,510	53.877	1.160
8	305,536				32	32				487	487	104.734	1.594
9	242,688				17	17				411	411	70.049	1.694
10	774,617				43	43				1,313	1,313	55.511	1.695
11	511,680				47	47				1,038	1,038	91.854	2.029
12	465,856			1	71	72			300	733	1,033	154.554	2.217
13	221,760				10	10				492	492	45,094	2.219
14	793,695			1	65	66			300	1,584	1,884	83.155	2.374
15	283,392			1	30	31			300	413	713	109.389	2.516
16	264,024				25	25				842	842	94.688	3.189
17	1,082,547			4	105	109			1,200	2,707	3,907	100.688	3.609
18	271,920			1	20	21			750	344	1,094	77.229	4.023
19	1,029,592			1	65	66			2,400	2,313	4,713	64.103	4.578
20	1,669,650	1		1	60	62	6,000		1,800	1,275	9,075	37.134	5.435
21	505,012			1	42	43			1,800	1,116	2,916	85.146	5.774
22	465,968			2	33	35			3,300	993	4,293	75.112	9.213
23	939,004	1		1	38	40	6,000		1,800	1,043	8,843	42,598	9.417
24	722,979		1		51	52		6,000		1,645	7,645	71.925	10.574
25	1,828,119	2	1		112	115	12,000	6,000		3,376	21,376	62.906	11.693
26	581,080	1			21	22	6,000			985	6,985	37,861	12.021
27	401,430	1			35	36	6,000			881	6,881	89.679	17.141
28	880,711	2			54	56	12,000			3,920	15,920	63.585	18.076
29	705,656	2			52	54	12,000			1,058	13,058	76.525	18.505
30	651,864	2			49	51	12,000			1,556	13,556	78.237	20.796
31	804,655	3			43	46	18,000			1,016	19,016	57.167	23.632
32	841,550	3			45	48	18,000			2,035	20,035	57.038	23.807
33	700,800	2		1	74	77	12,000		4,500	1,562	18,062	109.874	25.773
34	345,520	2			27	29	12,000			1,111	13,111	83.931	37.946
35	652,839	3	1		60	64	18,000	6,000		1,368	25,368	98.033	38.858
36	858,048	6		4	121	131	36,000		3900	2,297	42,197	152.672	49.178
37	188,136	3			20	23	18,000			631	18,631	122.252	99.029
Totals and averages, 1935	24,475,637	34	3	22	1,698	1,757	204,000	18,000	23,250	44,241	289,491	71.786	11.828
Totals and averages, 1934	16,526,073	19	1	16	1,275	1,311	114,000	6,000	18,000	30,251	168,251	79.329	10.181

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permenent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from accidonts per thousand man-hours.

TABLE 3.- Accident data for certain BITUMINOUS COAL MINES In 1935.[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
1	235,211											0.000	0.000
2	225,631											.000	.000
3	180,964				4	4				72	72	22.104	.398
4	157,052				3	3				93	93	19.102	.592
5	280,684				3	3				197	197	10.688	.702
6	226,460				2	2				176	176	8.832	.777
7	47,093				6	6				40	40	127.407	.849
8	708,932				15	15				831	831	21.159	1.172
9	488,189				19	19				597	597	38.919	1.223
10	182,162				27	27				231	231	148.220	1.268
11	111,005				11	11				143	143	99.095	1.288
12	16,218				1	1				21	21	61.660	1.295
13	135,076				4	4				175	175	29.613	1.296
14	139,504				6	6				182	182	43.010	1.305
15	209,834			1	2	3			300	6	306	14.297	1.458
16	365,534				23	23				544	544	62.922	1.488
17	197,857				6	6				306	306	30.325	1.547
18	76,843				10	10				123	123	130.135	1.601
19	1,441,569				34	34				2,357	2,357	23.585	1.635
20	294,386				17	17				500	500	57.747	1.698
21	644,005				24	24				1,169	1,169	37.267	1.815
22	83,286				21	21				155	155	252,143	1.861
23	111,782				8	8				210	210	71.568	1.879
24	333,452				24	24				657	657	71.974	1.970
25	229,816				11	11				472	472	47.864	2.054
26	509,593			1	7	8			600	448	1,048	15.699	2.057
27	553,138				60	60				1,157	1,157	108.472	2.092
28	439,005			1	7	8			200	750	950	18.223	2.164
29	777,868				25	25				1,769	1,769	32.139	2.274
30	256,377				9	9				590	590	35.105	2.301
31	898,153			3	11	14			1,650	450	2,100	15.588	2.338
32	54,600				10	10				129	129	183.150	2.363
33	359,744			1	41	42			300	671	971	116.750	2.699
34	376,234				34	34				1,048	1,048	90.369	2.786
35	181,800			1	9	10			300	220	520	55.006	2.860
36	129,020				6	6				373	373	46.504	2.891
37	282,891				23	23				821	821	81.303	2.902
38	221,084			1	3	4			600	52	652	18.093	2.949
39	127,426				6	6				391	391	47.086	3.068
40	380,374			1	19	20			300	878	1,178	52.580	3.097
41	107,730				8	8				336	336	74.260	3.119

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporaty disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure, severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 3.- Accident data for certain BITUMINOUS COAL MINES in 1935 - Continued[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Seferity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
42	290,619				30	30				934	934	103.228	3.214
43	217,679				19	19				706	706	87.284	3.243
44	346,448				15	15				1,131	1,131	43.297	3.265
45	96,758				18	18				324	324	186.031	3.349
46	710,017			2	113	115			1,050	1,627	2,677	161.968	3.770
47	158,053			1	8	9			300	329	629	56.943	3.980
48	145,591				26	26				531	531	178.582	3.991
49	153,862			1	2	3			300	318	618	19.498	4.017
50	359,575			1	23	24			750	706	1,456	66.745	4.049
51	418,565			2	55	57			600	1,166	1,766	136.180	4.219
52	282,779			1	6	7			300	922	1,222	24.754	4.321
53	312,004			1	31	32			300	1,089	1,389	102.563	4.452
54	246,752			2	4	6			960	232	1,192	24.316	4.831
55	560,457			2	29	31			1,940	850	2,790	55.312	4.978
56	200,289			1	14	15			300	701	1,001	74.892	4.998
57	110,089				5	5				573	573	45.418	5.205
58	1,043,238				31	31				5,469	5,469	29.715	5.242
59	180,666				15	15				951	951	83.026	5.264
60	777,401			1	22	23			3,000	1,147	4,147	29.586	5.334
61	373,647			1	18	19			1,200	838	2,038	50.850	5.454
62	354,208			3	7	10			1,800	138	1,938	28.232	5.471
63	67,573			1	5	6			300	102	402	88.793	5.949
64	546,182			2	19	21			2,100	1,259	3,359	38.449	6.150
65	509,219			1	36	37			750	2,517	3,267	72.660	6.416
66	93,891			1	5	6			600	74	674	63.904	7.179
67	856,500	1		2	21	24	6,000		1,050	702	7,752	28.021	9.051
68	778,758	1			74	75	6,000			1,117	7,117	96.307	9.139
69	762,239	1			30	31	6,000			1,335	7,335	40.670	9.623
70	180,030			4	23	27			1,260	637	1,897	149.975	10.537
71	622,300	1		1	56	58	6,000		300	1,135	7,435	93.203	11.948
72	756,887	1		3	28	32	6,000		2,250	906	9,156	42.278	12.097
73	401,381			2	6	8			4,720	272	4,992	19.931	12.437
74	1,581,172	3		1	14	18	18,000		1,200	715	19,915	11.384	12.595
75	511,237	1			19	20	6,000			443	6,443	39.121	12.603
76	634,116	1		2	10	13	6,000		2,100	400	8,500	20.501	13.404
77	562,801	1			15	16	6,000			1,839	7,839	28.429	13.929
78	522,525	1			48	49	6,000			1,389	7,389	93.775	14.141
79	467,600	1			42	43	6,000			807	6,807	91.959	14.557
80	458,982	1			5	6	6,000			774	6,774	13.072	14.759
81	438,184	1		2	15	18	6,000		600	360	6,960	41.079	15.884
82	466,105	1		3	11	15	6,000		900	567	7,467	32.182	16.020

1 As the accident reports from mining companies are considered confidential by the Bureau of Minos, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanont partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure, severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 3.- Accident data for certain BITUMINOUS COAL MINES in 1935 - Concluded[1]

	Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
			Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
	83	255,608			2	17	19			3,600	499	4,099	74.333	16.036
	84	746,956	1		2	20	23	6,000		4,800	1,647	12,447	30.792	16.664
	85	138,719			1	12	13			1,800	533	2,333	93.715	16.818
	86	955,885	1		12	26	39	6,000		10,283	863	17,146	40.800	17.937
	87	354,798	1			24	25	6,000			522	6,522	70.463	18.382
	88	358,750		1		55	56		6,000		829	6,829	156.098	19.036
	89	353,013	1		1	8	10	6,000		600	146	6,746	28.328	19.110
	90	471,940	1		1	19	21	6,000		3,000	514	9,514	44.497	20.159
	91	375,872	1		1	26	28	6,000		300	1,679	7,979	74.493	21.228
	92	329,538	1		1	18	20	6,000		600	529	7,129	60.691	21.633
	93	277,219	1				1	6,000				6,000	3.607	21.644
	94	291,112	1			13	14	6,000			516	6,516	48.091	22.383
	95	704,331	2		4	3	9	12,000		3,900	128	16,028	12.778	22.756
	96	266,515	1			2	3	6,000			110	6,110	11.256	22.926
	97	254,524	1			1	2	6,000			63	6,063	7.858	23.821
	98	273,618	1		4	36	41	6,000		430	973	7,403	149.844	27.056
	99	336,853	1		2	34	37	6,000		3,300	763	10,063	109.840	29.874
	100	611,434	2		2	54	58	12,000		5,100	1,704	18,804	94.859	30.754
	101	619,548	3		1	13	17	18,000		300	1,069	19,369	27.439	31.263
	102	196,203	1			12	13	6,000			356	6,356	66.258	32.395
	103	188,996	1			8	9	6,000			296	6,296	47.620	33.313
	104	353,610	1		2	26	29	6,000		7,875	795	14,670	82.011	41.486
	105	911,142	6		11	38	55	36,000		2,565	1,777	40,342	60.364	44.276
	106	66,780				1	1				3	3	14.975	44.924
	107	357,966	3		2	22	27	18,000		600	641	19,241	75.426	53.751
	108	352,704	3		2	22	27	18,000		450	1,072	19,522	76.551	55.350
	109	604,618	4		5	9	18	24,000		8,740	1.012	33,752	29.771	55.824
Totals and averages, 1935		42,342,213	55	1	110	2,051	2,217	330,000	6,000	93,423	76,411	505,834	52.359	11.946
Totals and averages, 1934		38,239,192	32	1	81	1,680	1,794	192,000	6,000	73,080	58,318	329,398	46.915	8.614

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure, severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 4.- Accident data for certain METAL MINES in 1935.[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
1	196,201											0.000	0.000
2	123,152											.000	.000
3	75,536											.000	.000
4	230,580				2	2				43	43	8.674	.186
5	372,803				2	2				77	77	5.365	.207
6	55,785				2	2				19	19	35.852	.341
7	220,593				2	2				151	151	9.066	.685
8	923,016			1	10	11			300	543	843	11.917	.913
9	163,239				3	3				157	157	18,378	.962
10	252,962				8	8				266	266	31.625	1.052
11	115,697				2	2				138	138	17.287	1.193
12	48,764				2	2				75	75	41.014	1.538
13	109,578				1	1				176	176	9.126	1.606
14	320,231			1	3	4			300	310	610	12.491	1.905
15	137,107			2		2			270		270	14.587	1.969
16	202,279				18	18				676	676	88.896	3.342
17	358,299				7	7				1,641	1,641	19.537	4.580
18	1,472,212	1			44	45	6,000			1,319	7,319	30.566	4.971
19	1,826,780	3		2	96	101	18,000		900	1,296	20,196	55.289	11.056
20	532,330	1		1	10	12	6,000		300	623	6,923	22.542	13.005
21	336,088	1			4	5	6,000			133	6,133	14,877	18.248
22	456,230	1		1	7	9	6,000		3,000	279	9,279	19.727	20.338
23	255,793		1			1		6,000			6,000	3.909	23.456
24	213,929	1			8	9	6,000			134	6,134	42.070	28.673
Total and averages, 1935	8,999,184	8	1	8	231	248	48,000	6,000	5,070	8,056	67,126	27.558	7.459
Totals and averages, 1935	10,214,228	5	1	10	318	334	30,000	6,000	8,210	11,415	55,625	32.699	5.446

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 5.- Accident data for certain NONMETALLIC MINES in 1935[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
1	87,271											0.000	0.000
2	79,144											.000	.000
3	47,575											.000	.000
4	46,632											.000	.000
5	46,128											.000	.000
6	37,852											.000	.000
7	37,699											.000	.000
8	19,200											.000	.000
9	344,875				1	1				16	16	2.900	.046
10	30,613				1	1				6	6	32.666	.196
11	100,000				2	2				29	29	20,000	.290
12	346,100				2	2				166	166	5.779	.480
13	60,361				1	1				30	30	16.567	.497
14	77,045				1	1				45	45	12.979	.584
15	281,109				5	5				222	222	17.787	.790
16	87,856				1	1				73	73	11.382	.831
17	38,622				4	4				56	56	103.568	1.450
18	65,269				2	2				104	104	30.642	1.593
19	98,949				2	2				204	204	20.212	2.062
20	52,240				1	1				194	194	19.142	3.714
Totals and averages, 1935	1,984,540	0	0	0	23	23	0	0	0	1,145	1,145	11,590	.577
Totals and averages, 1934	1,935,930	3	0	1	42	46	18,000	0	270	1,128	19,398	23.761	10.020

1 As the accident reports from mining companies are considered confidental by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., oermanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 6.- Accident data for certain QUARRIES AND OPENCUT MINES in 1935[1]

Group no.	Hours worked	Number of accidents[2]					Number of days lost[2]					Frequency rate[3]	Severity rate[3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P. P.	Temp.	Total		
1	717,168											0.000	.000
2	258,894											.000	.000
3	198,398											.000	.000
4	175,086											.000	.000
5	167,671											.000	.000
6	129,336											.000	.000
7	122,934											.000	.000
8	120,733											.000	.000
9	118,747											.000	.000
10	114,732											.000	.000
11	114,632											.000	.000
12	111,928											.000	.000
13	107,263											.000	.000
14	105,598											.000	.000
15	98,953											.000	.000
16	95,061											.000	.000
17	91,399											.000	.000
18	87,546											.000	.000
19	87,370											.000	.000
20	86,386											.000	.000
21	84,869											.000	.000
22	84,396											.000	.000
23	79,140											.000	.000
24	75,967											.000	.000
25	69,811											.000	.000
26	65,268											.000	.000
27	62,899											.000	.000
28	57,982											.000	.000
29	57,012											.000	.000
30	53,835											.000	.000
31	53,487											.000	.000
32	53,201											.000	.000
33	52,575											.000	.000
34	51,808											.000	.000
35	51,403											.000	.000
36	50,299											.000	.000
37	49,270											.000	.000
38	47,421											.000	.000
39	47,278											.000	.000
40	46,953											.000	.000

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from acoidents per thousand man-hours.

TABLE 6.- Accident data for certain QUARRIES AND OPENCUT MINES in 1935 - Continued [1]

Group no.	Hours worked	Number of accidents [2]					Number of days lost [2]					Frequency rate [3]	Severity rate [3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
41	46,790											.000	.000
42	46,626											.000	.000
43	46,626											.000	.000
44	45,079											.000	.000
45	45,054											.000	.000
46	44,297											.000	.000
47	42,864											.000	.000
48	42,491											.000	.000
49	39,984											.000	.000
50	39,841											.000	.000
51	39,277											.000	.000
52	38,079											.000	.000
53	35,973											.000	.000
54	35,685											.000	.000
55	35,097											.000	.000
56	34,974											.000	.000
57	33,558											.000	.000
58	32,130											.000	.000
59	30,531											.000	.000
60	30,478											.000	.000
61	30,129											.000	.000
62	30,077											.000	.000
63	29,003											.000	.000
64	27,602											.000	.000
65	26,978											.000	.000
66	26,234											.000	.000
67	26,074											.000	.000
68	23,100											.000	.000
69	22,363											.000	.000
70	21,873											.000	.000
71	21,160											.000	.000
72	20,657											.000	.000
73	20,132											.000	.000
74	19,299											.000	.000
75	18,557											.000	.000
76	17,393											.000	.000
77	16,769											.000	.000
78	16,325											.000	.000
79	15,444											.000	.000
80	15,374											.000	.000

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 6.- Accident data for certain QUARRIES AND OPENCUT MINES in 1935 - Continued [1]

Group no.	Hours worked	Number of accidents [2]					Number of days lost [2]					Frequency rate [3]	Severity rate [3]
		Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total		
81	15,086											0.000	0.000
82	11,371											.000	.000
83	10,393											.000	.000
84	761,368				1	1				14	14	1.313	.018
85	107,442				1	1				7	7	9.307	.065
86	99,867				2	2				8	8	20.027	.080
87	766,351				1	1				67	67	1.305	.087
88	50,411				1	1				5	5	19.837	.099
89	138,466				1	1				15	15	7.222	.108
90	64,992				1	1				9	9	15.387	.138
91	392,278				4	4				56	56	10.197	.143
92	146,202				3	3				21	21	20.520	.144
93	241,036				3	3				39	39	12.446	.162
94	48,418				1	1				10	10	20.653	.207
95	130,445				4	4				44	44	30.664	.337
96	13,440				1	1				5	5	74.405	.372
97	1,016,002			1	8	9			300	91	391	8.858	.385
98	69,008				5	5				28	28	72.455	.406
99	67,200				2	2				28	28	29.762	.417
100	27,742				2	2				16	16	72,093	.577
101	136,792				4	4				89	89	29.241	.651
102	72,326				1	1				49	49	13.826	.677
103	298,664				9	9				214	214	30.134	.717
104	73,441				1	1				62	62	13.616	.844
105	38,852				1	1				33	33	25.739	.849
106	78,266				1	1				72	72	12.777	.920
107	44,609				3	3				42	42	67.251	.942
108	58,063				2	2				57	57	34.445	.982
109	155,064				14	14				160	160	90.285	1.032
110	63,359				2	2				66	66	31.566	1.042
111	31,749				4	4				35	35	125.988	1.102
112	31,424				1	1				35	35	31.823	1.114
113	62,071				3	3				75	75	48.332	1.208
114	307,619				14	14				397	397	45.511	1.291
115	43,632				1	1				58	58	22.919	1.329
116	80,725				1	1				117	117	12.388	1.449
117	81,930				1	1				123	123	12.206	1.501
118	26,844				1	1				41	41	37.252	1.527
119	90,342				11	11				154	154	121.760	1.705
120	52,004				4	4				90	90	76.917	1.731

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates the number days lost from accidents per thousand man-hours.

TABLE 6.- Accident data for certain QUARRIES AND OPENCUT MINES in 1935 - Concluded[1]

	Group	Hours	Number of accidents[2]					Number of days lost[2]					Frequency	Severity
	no.	worked	Fatal	P.T.	P.P.	Temp.	Total	Fatal	P.T.	P.P.	Temp.	Total	rate[3]	rate[3]
	121	87,356	.....	.....	.....	5	5	.....	.....	.....	155	155	57.237	1.774
	122	96,475	.....	.....	.....	2	2	.....	.....	.....	238	238	20.731	2.467
	123	38,161	.....	.....	.....	1	1	.....	.....	.....	98	98	26.205	2.568
	124	33,894	.....	.....	.....	1	1	.....	.....	.....	90	90	29.504	2.655
	125	39,389	.....	.....	1	.....	1	.....	.....	105	.....	105	25.388	2.666
	126	111,867	.....	.....	1	.....	1	.....	.....	300	.....	300	8.939	2.682
	127	32,937	.....	.....	.....	1	1	.....	.....	.....	95	95	30,361	2.884
	128	44,980	.....	.....	.....	1	1	.....	.....	.....	145	145	22.232	3.224
	129	434,361	.....	.....	1	1	2	.....	.....	1,500	49	1,549	4.604	3.566
	130	70,283	.....	.....	1	.....	1	.....	.....	300	.....	300	14.228	4.268
	131	33,705	.....	.....	.....	1	1	.....	.....	.....	163	163	29.669	4.636
	132	62,142	.....	.....	1	.....	1	.....	.....	300	.....	300	16.092	4.828
	133	66,753	.....	.....	1	1	2	.....	.....	300	26	326	29.961	4.884
	134	60,345	.....	.....	1	.....	1	.....	.....	300	.....	300	16.571	4.971
	135	44,822	.....	.....	1	.....	1	.....	.....	300	.....	300	22.310	6.693
	136	42,622	.....	.....	1	.....	1	.....	.....	300	.....	300	23.462	7.039
	137	41,827	.....	.....	1	1	2	.....	.....	300	61	361	47.816	8.631
	138	21,780	.....	.....	.....	3	3	.....	.....	.....	207	207	137.741	9.504
	139	100,558	.....	.....	1	.....	1	.....	.....	1,800	.....	1,800	9.945	17.900
	140	146,472	.....	.....	1	6	7	.....	.....	3,000	74	3,074	47.791	20.987
	141	473,660	1	1	.....	3	5	6,000	6,000	.....	1,082	13,082	10.556	27.619
	142	200,353	1	.....	.....	7	8	6,000	.....	.....	282	6,282	39.930	31.355
	143	137,189	1	.....	.....	8	9	6,000	.....	.....	284	6,284	65.603	45.805
	144	131,047	1	.....	.....	6	7	6,000	.....	.....	110	6,110	53,416	46,624
	145	40,985	.....	.....	1	1	2	.....	.....	2,400	39	2,439	48.798	59.510
	146	43,357	.....	.....	1	11	12	.....	.....	4,500	129	4,629	276.772	106.765
Totals and averages, 1935		14,177,300	4	1	15	180	200	24,000	6,000	16,005	5,759	51,764	14.107	3.651
Totals and averages, 1934		14,844,434	7	0	11	231	249	42,000	0	9,545	4,688	56,233	16.774	3.788

1 As the accident reports from mining companies are considered confidential by the Bureau of Mines, the identities of the mines to which this table relates are not revealed.

2 P. T., permanent total disability; P. P., permanent partial disability; Temp., temporary disability.

3 Frequency rate indicates the number of fatal, permanent, and other lost-time accidents per million man-hours of exposure; severity rate indicates number of days lost from accidents per thousand man-hours.

TABLE 7.- Accident-frequency rates for mines and quarries in the National Safety Competition[1]

Type of mine	Type of injury[2]	Year										
		1925	1926	1927	1928	1929	1930	1931	1932	1933	1934	1935
Anthracite	Fatal	1.022	0.837	1.253	1.401	1.350	1.186	1.273	1.387	0.998	1.150	1.389
	P.T.	.000	.000	.000	.000	.000	.108	.064	.060	.130	.060	.123
	P.P.	.205	.963	.861	1.138	.675	.934	.828	.482	1.215	.968	.899
	Temp.	93.450	96.146	97.438	85.577	99.330	107.111	82.458	99.467	101.570	77.151	69.375
	Total	94.677	97.946	99.552	88.116	101.355	109.339	84.623	101.396	103.913	79.329	71.786
Bituminous	Fatal	1.156	1.286	1.915	1.399	1.247	1.785	1.093	1.017	1.026	.837	1.299
	P.T.	.041	.038	.091	.034	.119	.038	.068	.000	.029	.026	.023
	P.P.	1.114	.832	2.097	2.184	1.812	1.291	2.767	2.099	2.433	2.118	2.598
	Temp.	126.015	65.478	93.223	81.342	65.759	72.571	55.195	49.103	44.411	43.934	48.439
	Total	128.326	67.634	97.326	84.959	68.937	75.685	59.123	52.219	47.899	46.915	52.359
Metal	Fatal	1.027	.843	.891	1.271	.497	.593	.878	.711	1.169	.489	.889
	P.T.	.000	.000	.000	.000	.055	.059	.160	.285	.000	.098	.111
	P.P.	.616	.649	1.337	.969	1.711	1.187	1.677	1.280	.584	.979	.889
	Temp.	133.446	113.944	61.673	49.883	49.899	17.623	30.497	35.137	37.846	31.133	25.669
	Total	135.089	115.436	63.901	52.123	52.162	19.462	33.212	37.413	39.599	32.699	27.558
Nonmetal	Fatal	.881	.772	.371	1.026	1.026	.587	1.255	.663	.000	1.550	.000
	P.T.	.000	.000	.000	.000	.000	.000	.000	.000	.000	.000	.000
	P.P.	1.174	.772	1.112	1.369	1.283	.880	.836	.000	.709	.516	.000
	Temp.	78.660	77.400	31.896	50.294	28.740	33.443	14.641	13.271	22.688	21.695	11.590
	Total	80.715	78.944	33.379	52.689	31.049	34.910	16.732	13.934	23.397	23.761	11.590
Quarry and opencut	Fatal	.799	.904	.576	.645	.861	.452	.509	.481	.671	.472	.282
	P.T.	.000	.118	.000	.000	.000	.000	.000	.000	.074	.000	.071
	P.P.	.940	.668	.615	.646	.709	.769	1.069	1.283	.969	.741	1.058
	Temp.	46.520	41.551	32.346	28.136	25.212	20.852	20.361	19.969	21.533	15.561	12.696
	Total	48.259	43.241	33.537	29.327	26.782	22.073	21.939	21.733	23.247	16.774	14.107
Grand total	Fatal	.994	.979	1.131	1.199	1.070	1.056	.955	.969	.951	.807	1.098
	P.T.	.015	.042	.022	.011	.047	.052	.063	.044	.063	.037	.065
	P.P.	.891	.789	1.175	1.319	1.239	1.036	1.735	1.425	1.637	1.455	1.685
	Temp.	98.190	75.124	69.963	64.056	63.990	59.710	46.867	53.803	56.263	43.371	45.478
	Total	100.090	76.934	72.291	66.585	66.346	61.854	49.620	56.241	58.914	45.670	48.326

1 Accident-frequency rate indicates the number of injuries and deaths per million man-hours of exposure.

2 P.T., permanent total disability; P.P., permanent partial disability; Temp., temporary disability.

TABLE 8.- Accident-severity rates for mines and quarries in the National Safety Competition[1]

Type of mine	Type of injury[2]	Year									
		1925	1926	1927	1928	1929	1930	1931	1932	1933	1934	1935
Anthracite	Fatal	6.134	5.025	7.516	8.404	8.103	7.114	7.461	8.319	5.990	6.898	8.335
	P.T.	.000	.000	.000	.000	.000	.647	.382	.362	.781	.363	.735
	P.P.	.123	.938	.781	.742	.661	1.082	.668	.271	1.394	1.089	.950
	Temp.	1.387	1.501	1.830	1.772	1.801	1.942	1.342	2.015	2.308	1.831	1.808
	Total	7.644	7.464	9.927	10.918	10.565	10.785	10.033	10.967	10.473	10.181	11.828
Bituminous	Fatal	6.935	7.717	11.488	8.393	7.485	10.709	6.558	6.101	6.156	5.021	7.794
	P.T.	.248	.227	.547	.205	.713	.228	.410	.000	.176	.157	.142
	P.P.	1.300	1.163	1.748	1.742	1.566	.903	2.030	1.361	1.753	1.911	2.206
	Temp.	2.003	1.600	1.956	2.131	1.807	1.860	1.484	1.566	1.355	1.525	1.804
	Total	10.486	10.707	15.739	12.471	11.571	13.700	10.482	9.028	9.440	8.614	11.946
Metal	Fatal	6.159	5.061	5.346	7.628	2.981	3.560	5.269	4.267	7.014	2.937	5.334
	P.T.	.000	.000	.000	.000	.331	.356	.958	1.707	.000	.587	.667
	P.P.	.616	.689	1.717	.999	1.635	.781	1.593	1.558	.175	.804	.563
	Temp.	1.887	1.769	1.331	.884	1.043	.585	1.029	1.523	1.219	1.118	.895
	Total	8.662	7.519	8.394	9.511	5.990	5.282	8.849	9.055	8.408	5.446	7.459
Nonmetal	Fatal	5.283	4.629	2.225	6.159	6.158	3.520	7.530	3.981	.000	9.298	.000
	P.T.	.000	.000	.000	.000	.000	.000	.000	.000	.000	.000	.000
	P.P.	.925	.308	1.335	.985	1.014	.389	.460	.000	.212	.139	.000
	Temp.	1.442	1.466	.853	.791	.622	.989	.451	.422	.756	.583	.577
	Total	7.650	6.403	4.413	7.935	7.794	4.898	8.441	4.403	.968	10.020	.577
Quarry and opencut	Fatal	4.793	5.425	3.458	3.871	5.164	2.714	3.054	2.887	4.024	2.829	1.693
	P.T.	.000	.707	.000	.000	.000	.000	.000	.000	447	.000	.423
	P.P.	.969	.914	.726	.774	.925	.683	1.401	1.249	.945	.643	1..129
	Temp.	.668	.714	.557	.563	.458	.364	.463	.516	.498	.316	.406
	Total	6.430	7.760	4.741	5.208	6.547	3.761	4.918	4.652	5.914	3.988	3.651
Grand total	Fatal	5.963	5.870	6.788	7.194	6.420	6.340	5.733	5.817	5.709	4.843	6.589
	P.T.	.088	.253	.131	.065	.282	.311	.377	.264	.380	.220	.391
	P.P.	.948	.928	1.172	1.123	1.174	.865	1.490	1.066	1.346	1.335	1.498
	Temp.	1.491	1.360	1.330	1.430	1.382	1.287	1.101	1.453	1.465	1.294	1.474
	Total	8.490	8.411	9.421	9.812	9.258	8.803	8.701	8.600	8.900	7.692	9.952

1 Accident-severity rate indicates the number of days of disability per thousand man-hours of exposure.

2 P.T., permanent total disability; P.P., permanent partial disability; Temp., temporary disability.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

I22.09
un32mo
Geo2

R. I. 3309 August 1936

DEPARTMENT OF THE INTERIOR
UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

PERMISSIBLE ELECTRICALLY OPERATED AIR COMPRESSORS

BY

L. C. ILSLEY, E. J. GLEIM, AND H. B. BRUNOT

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3309,
August 1936.

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

PERMISSIBLE ELECTRICALLY OPERATED AIR COMPRESSORS[1]

By L. C. Ilsley,[2] E. J. Gleim,[3] and H. B. Brunot[4]

INTRODUCTION

One of the various phases of the work by the U.S. Bureau of Mines, in its endeavors to minimize the hazards that attend coal mining, is the investigation of electrical motor-driven machinery. The object of this investigation is to foster the development of explosion-proof machines for use in gassy mines. This investigation is conducted under the provisions of Schedule 2C, which prescribes the inspection and tests made upon motors, controllers, and other electrical parts to determine their ability to withstand internal explosion without discharge of any flames. The method of conducting these inspections and tests is described in Bureau of Mines Bulletin 305, Inspection and Testing of Mine-Type Electrical Equipment for Permissibility, published in 1929. Formal approval of a machine for use in gassy mines is not granted by the Bureau of Mines until the machine is found to comply with the requirements, including inspection and test. When it is approved it is entitled to classification as a "permissible" machine.

The high regard in which machines approved by the U.S. Bureau of Mines are held by engineers of other countries is attested by the growing number of shipments of permissible equipment to other nations, including Australia, Russia, Norway, Japan, and Manchuria.

Many of the permissible machines come under one of three large groups, namely: (1) Storage-battery locomotives and power trucks; (2) coal-handling equipment, and (3) coal-cutting equipment. The first group includes locomotives of both the haulage and gathering type, as well as those that may be used as a source of energy for the operation of cutting machines, compressors, pumps, and any other portable machines for which it may not be safe or feasible to install feeder cables in the mine. This group has been described in Bureau

---

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3309."

2/ Electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3/ Electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

4/ Associate electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

of Mines Bulletin 313, Permissible Storage-Battery Locomotives and Power Trucks, published in 1929. In the second group are loading and shoveling machines, pit-car loaders, and conveyors. A detailed description of these is to be found in Bureau of Mines Bulletin 343, Permissible Coal-Handling Equipment, published in 1931. The third group comprises the various shortwall, longwall, turret, and other types of cutting machines. These are described in Bureau of Mines Bulletin 382, Permissible Coal-Cutting Equipment, published in 1934.

Another group of machines not included in the bulletins just mentioned has never been described at length in Bureau publications. This fourth group includes miscellaneous equipment, which, although not as widely used as some in the other groups, is nevertheless just as important from the standpoint of safety. It includes the air compressors, pumps, room hoists, rock-dust distributors, and drills. The air compressors approved prior to May 1, 1936, are described in the present paper.

In certain kinds of work in coal mines, such as driving entries through rock, it is often advantageous to use jackhammers and other pneumatic tools. The portable air compressor therefore becomes a useful piece of equipment in an electrified mine, since it would not be economical to install and maintain a compressor station and long pipe lines to supply the comparatively small amounts of air required. When an electrically driven compressor is required in a gassy mine, the permissible type, because of its special construction from the standpoint of safety, should be given first consideration. There are now six approved portable compressors in the Bureau's List of Permissible Mine Equipment. These represent the product of three different manufacturers. All of these machines are mounted on trucks suitable for transportation about the mines. Only two are built to be self-propelling, and only two have cable reels.

DETAILED DESCRIPTIONS

Sullivan Machinery Co. Type WK-26 Compressor
Approvals Nos. 117 and 117A

The Sullivan Machinery Co. type WK-26 compressor was approved for both 250- and 500-volt direct-current service on March 12, 1925. Approval No. 117 identifies the 250-volt machine and Approval No. 117A identifies the 500-volt machine.

The air receiver, motor, and control units of the WK-26 compressor outfit are mounted on one side of the truck and the flywheel and compressor proper are on the opposite side.

Current for operating the motor is received through a trailing cable, which is connected to insulated studs passing through the wall of the control-unit cover. The control unit contains a single-pole circuit breaker, which serves also as a switch and which can be tripped open or reset from the outside of the compartment by means of a knob. The unit also contained a starting rheostat and a drum controller operated by a handle. The control unit is

bolted directly to the side of the motor, and electrical connections between the two are made by means of insulated studs.

In subsequent modifications of the machine covered by extensions of the original approval, the substitution of a combination main-line switch and fuse for the circuit breaker was included. This change consists of a 2-pole, single-throw, knife-blade switch and a single cartridge fuse enclosed in a housing bolted to the outside of the control unit. In this arrangement the fuse is renewable from the outside; consequently, an interlocking mechanism is necessary to prevent removal of the fuse plug while the switch is closed and also to lock the switch in the open position once the fuse plug is withdrawn. Connections between this compartment and the control unit are made with insulated studs.

The constructional features that distinguish the approved from the unapproved machine are the explosion-proof enclosures for the electrical parts. Strong casings with broad, machined surfaces at the joints protect against the discharge of flame should an explosion occur within them.

The motor used on the Sullivan WK-26 air compressor is built by the General Electric Co. It is designated as a special type CY-24 and has a rating of 30 horsepower. In some respects this motor resembles the one used on the Sullivan CE-7 shortwall mining machine, but differs from the latter in that solid instead of ventilated end heads are used.

The magnet frame of the special CY-24 motor is approximately rectangular in section. At the commutator end of the frame there is an inspection opening, which is normally closed by an oblong cast plate. This plate and the frame have flat, machined surfaces in contact, and the joint is made tight by six studs with nuts and lock washers. Originally, an optional plate with two circular openings through it, which were threaded for 6-inch screw covers, also was specified. These screw covers were locked together with a padlock and made it possible to inspect the commutator and brushes without removing the plate itself.

The rear motor head (pinion end) is secured to the motor frame by 10 cap screws. The joint between the frame and head consists, in the main, of flat, machined surfaces. At two points along the top and bottom of the frame where the flat surfaces are less than 1 inch wide, the lip that is turned on the head to center it in the frame gives the necessary additional width of joint to make a total of at least 1 inch. This head carries a ball bearing for the armature shaft. At both ends of the motor, obstruction to the passage of flame is achieved by close fit of the bearing housings upon the armature shaft, together with the resistance offered by the bearings.

The front head (commutator end) is held to the motor frame by eight cap screws. This head has circular lips for centering it in the frame, but these are not needed to give the necessary width of joint as was the case with the other head. The front head also carries a ball bearing for the armature shaft.

The controller on the machine as first approved was housed in a boxlike casting, with a deep ribbed cover fastened to it by 14 cap screws. The joint between the cover and box consisted of broad, flat, machined surfaces. Two rods or shafts passed through the right hand end of the box. One of these terminated in a knob, the other in a controller handle. When the former was pulled out with the controller in the "off" position, it locked the controller in that position. The locking rod passed through a reamed hole 2 inches long in the box and fitted the hole tightly enough to prevent the passage of flame. Similarly, the controller shaft passed through another reamed hole 2 inches long, making a tight fit that prevented flame passage. The circuit-breaker reset rod passed through a reamed hole 3 inches long in the cover of the controller box. The length of the hole and the fit of the rod in it were such that no flames could escape.

Two insulated studs in the cover provided for connecting the trailing cable with the machine. The external ends of these studs were housed in a junction box cast integral with the cover. The lid for the junction box had a recess on its underside to accommodate a split clamp of insulating material. The function of the clamp was to grip the cable so that pulling on it would not tend to loosen the connections on the studs.

When the circuit breaker was eliminated, the design of lead entrance just described was retained, but several changes were made in the large cover for the controller. It was made somewhat shallower than before; short internal ribs at the sides and ends replaced the external transverse stiffening ribs, and finally an area was machined on the outside against which to bolt the switch and fuse enclosure.

The housing for the switch and fuse is held to the controller cover by 10 cap screws secured with lock washers. The joint between these two castings is made up of flat, machined surfaces in close contact. The fuse is enclosed in a cartridge fastened to a plug terminating in an insulating knob, by which the whole can be handled. The plug portion originally was fitted with a sleeve and key of steel. Subsequently these parts were made of brass because rusting of the steel parts frequently made it difficult to remove the fuse plug without breaking the knob. To prevent the escape of flame around the fuse plug the sleeve and key had to be made to fit closely in the opening through the housing for the switch and fuse. When in the closed position, the switch-operating shaft engages a notch in the key and thus locks the fuse plug in place. The switch shaft passes through a hole more than 4 inches long. The length and size of the clearance around the shaft are such as to prevent the escape of flame.

Electrical connection between the switch and fuse compartment and the controller compartment was first effected by means of insulated studs. Later, the insulated studs were eliminated, and insulating bushings with closely fitting cables through them were used in their stead. In this manner, one long cable replaced two shorter ones. At the same time, cables in bushings also were substituted for the studs between the motor and controller compartments.

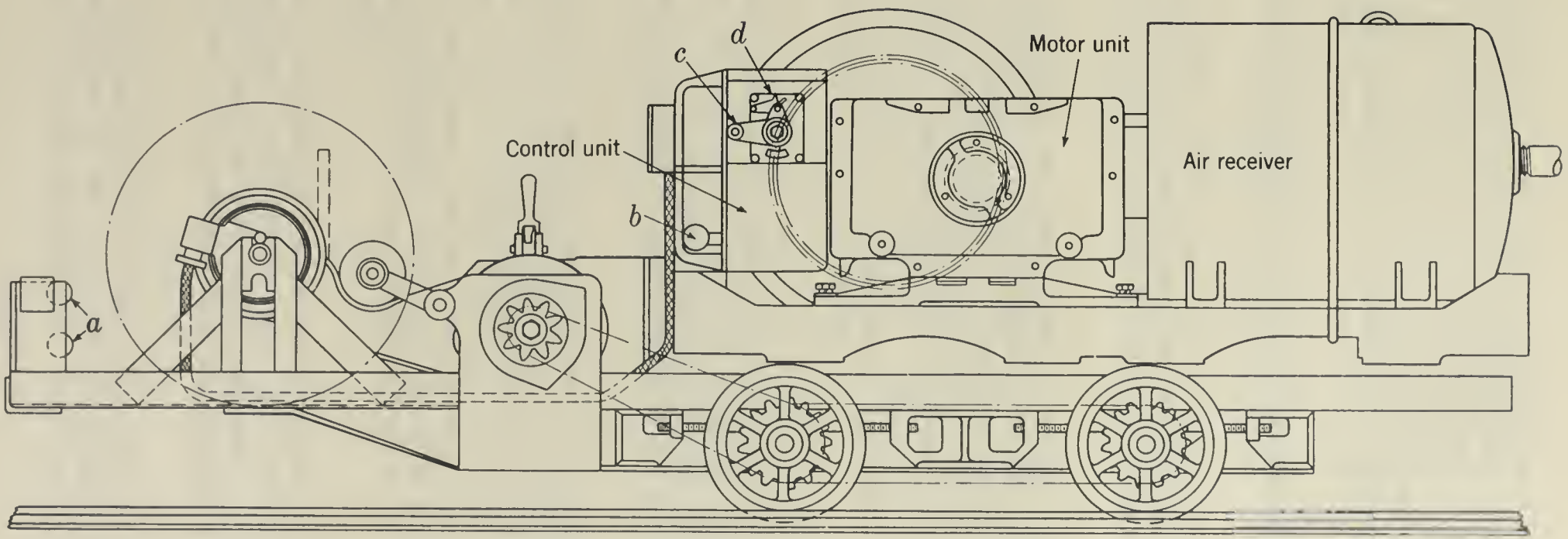


Figure 1.—Sullivan type WK–39 compressor.

The trailing cable first used with this compressor consisted of 450 feet of No. 4 concentric all-rubber insulated cable fitted with a rail clamp and fused trolley tap for connecting to the mine trolley circuit. Later, when a number of cables had successfully passed special tests designed to bring about the development of cables highly resistant to mechanical injury, the Sullivan Machinery Co. was given permission, by extension of approval, to use such "specially recommended" twin cables as were listed by the Bureau at the time.

Sullivan Machinery Co. Type WK-39 Compressor
Approvals Nos. 120 and 120A

The Sullivan type WK-39 self-propelled air compressor was approved for both 250- and 500-volt direct-current service on July 28, 1925. Approval No. 120 was assigned to the 250-volt compressor and Approval No. 120-A to the 500-volt compressor.

A view of the type WK-39 self-propelled air compressor as originally approved is shown in figure 1. The arrangement of this compressor equipment on the mine truck is the same as that for the type WK-26 compressor previously described. The truck of the type WK-39 compressor, however, is made longer than the former in order to accommodate the cable reel and other parts necessary to provide the self-propelling feature. The drive is effected through sprockets and chains. A clutch and a brake are included for controlling the movements of the compressor along the track.

Electric current for this compressor is received through a trailing cable that is fed in and out through the rollers a and wound on the cable reel. A slip-ring compartment at each end of the reel provides for making connection between the reel and the control unit. The cable reel itself is driven by a belt from a pulley on the jackshaft, which is part of the driving mechanism for the machine as a whole.

The first control unit for this machine was virtually a duplicate of that first used on the Sullivan type WK-26 compressor. It contained a circuit breaker operated by a shaft that terminated in the knob b, a starting rheostat, and a drum controller operated by the handle c. The essential difference between the two control units was that the controller proper on the type WK-39 machine was of the reversing type. This unit was also bolted directly to the side of the motor, and electrical connections between the two were made by means of insulated studs.

At the time when the circuit breaker was eliminated from the type WK-26 compressor and replaced by a combination main-line switch and fuse, an equivalent change also was made on the type WK-39 compressor. In addition, a device to enable the controller to be operated from the opposite side also was included.

A still later development was the change (on 250-volt machines) from a manually operated controller to one that was automatic. The main-line switch and fuse was retained with the new control. The automatic controller includes

a low voltage relay, an overload relay, and a series-interlocking relay. The first of these relays prevents operation of the compressor unless the voltage is above 75 percent of normal; and in case the voltage falls below 65 percent of normal the relay will stop the motor. The second relay protects the motor against excessive overloads. The purpose of the third relay is to afford protection to the motor in case the cable became disconnected while the motor was running. In such an event this relay restores the control circuits to the condition for starting and then if the cable is reconnected before the motor has stopped, full potential will not be applied to the armature.

Another extension of approval permitted a change in the material for the sleeve and key of the fuse plug in the main-line switch and fuse. At first these parts were made of steel, but, as explained in connection with the description of the WK-26 compressor, later they were made of brass.

The construction of explosion-proof enclosures for the first motor and controller of the Sullivan type WK-39 compressor was almost identical with that of the enclosures already described in connection with their type WK-26 machine. From time to time, however, important changes were made in the various electrical parts. A description covering the different stages in the evolution of the WK-39 compressor up to the present date follows:

The electrical connections between the motor and controller as well as between the controller and the switch and fuse compartment were first made by means of insulated studs. Subsequently the insulated studs were removed, and continuous lengths of cable run through insulating bushings replaced them. This enabled each connection to be made with one continuous length of cable instead of the two pieces formerly required.

Finally, when the automatic controller superseded the drum-type controller, another change was made in the method of making connection with the motor. The previous connections made use of six insulated studs or the same number of bushings. The new arrangement consists of two 3-conductor cables, each passing through a stuffing box. These stuffing boxes are put in a different place from that occupied by the bushings and studs.

As previously stated, there was little difference between the controllers as first used on the two types of Sullivan compressors. On the type WK-39 compressor the locking rod was omitted, and the hole for it was not drilled in the casing. Plate d (fig. 1), carrying a stop to prevent too sudden reversal of the motor, was riveted to the casing, and the hole for the controller shaft was thereby increased 13/16-inch in length over that of the type WK-26 compressor.

On the latter machine the trailing cable was connected to terminals of two insulated studs in the controller-box cover. On the self-propelled machine, however, the small cover for the connection box was altered to secure the ends of two flexible metallic conduits instead of holding an insulated clamp that gripped the trailing cable.

With the substitution of the main-line switch and fuse for the circuit breaker, the controller cover for the WK-39 compressor underwent the same changes in design as for the WK-26 compressor. However, when the contactor or magnetic controller displaced the drum-type controller on the former machine the enclosure itself underwent some important changes: The box was made longer and deeper. The greater part of the cover was shallower, but in one corner a pocket was made for enclosing a pilot and reversing switch. The method of fastening the cover was changed by using studs instead of cap screws and increasing the number used. The joint between the box and cover is still made up of flat machined surfaces in contact but is slightly narrower than formerly. The main-line switch and fuse box is fastened in the same manner and in the same relative position as before. The connections between the controller and switch compartments are made by carrying leads through insulating bushings. The connections between the controller and motor, as previously stated, are made by means of two 3-conductor cables passing through stuffing boxes. The cable external to the two compartments is inclosed in two air-hose conduits. The position of the lead entrance for the cables between the controller and the cable reel is changed from the left to the right side of the controller cover. Individual stuffing boxes are provided for each of the two conductors. These two cables also are given mechanical protection, but flexible metallic tubing is used for the purpose.

Two slip-ring housings, one at each end of the reel, comprise the explosion-proof features of the cable reel. Each housing is made up of two main parts, one of which remains stationary with the framework supporting the reel. The other part turns with the reel proper as the cable is wound and unwound. The joint between the two is made up of a cylindrical turned surface on the stationary part fitting closely within a cylindrical surface bored in the revolving part. Concentric with the cylindrical surface is a bored hole in each part of the shaft on which the reel rotates. The length and diameter of the hole are such that no flames can escape should there be ignition of gas in the slip-ring housings. The shaft and revolving part of the housing are held together by a steel pin. The stationary part of the housing is kept from moving out of the rotating part by a collar held to the shaft by means of a pin. An insulated stud through the wall of each rotating part provides for connection of one of the trailing-cable conductors. This stud connects with a ring that rubs against a similar ring in the stationary member, and the electrical circuit is continued from thence to the controller through another insulated stud that passes through the wall of the stationary member.

When the approval of the type WK-39 compressor was issued it covered a 450-foot length of No. 4 concentric, all-rubber, insulated trailing cable with a rail clamp and fused trolley tap for connecting to the power circuits in the mine. An extension of approval gave the Sullivan Machinery Co. authority to use a number of the cables specially recommended by the Bureau.

General Electric Co. Types CP-26D, G & H Compressors
Approvals Nos. 128 and 128A

The approved air compressors built by the General Electric Co. are made in three different types, CP-26D, CP-26G, and CP-26H. These types denote certain variations in the general design. Figure 2 shows the type CP-26H machine.

The General Electric Co. type CP-26D and H (500-volt) compressors were officially approved on July 16, 1926. The specifications for the 250-volt type CP-26G machine were not completed until after this date and consequently approval of the latter machine was not granted until March 21, 1927. The 250-volt compressor is identified by Approval No. 128, and No. 128A covers the 500-volt design.

The compressors have four cylinders for two-stage operation and are driven by a series-wound direct-current motor having a rating of 25 horsepower. A governor actuated by the pressure in the air receiver allows the motor to start when the pressure is below the predetermined value by closing the control circuit. When the pressure exceeds the desired value the governor opens the control circuit, stopping the motor.

Overload protection for the motor is afforded by a thermal relay with an element for each side of the main circuit to open the control circuit when the load on the motor becomes excessive. Short-circuit protection was not included at first, but later this feature was required and a fuse was added to meet the requirement.

No starting resistor is used in connection with the operation of the motor. A master or starting switch closes or opens the control circuit and a contactor, or contactors, is actuated, thus starting or stopping the motor. The handle of this switch is shown at a, figure 2.

The box for the master switch is equipped with a socket that receives a plug, b, attached to the trailing cable. The handle of the master switch is interlocked mechanically with the plug so that the latter can be neither inserted nor withdrawn unless the master switch is open. It is possible, however, to have live contacts in the plug after it is withdrawn. The original approval did not interfere with promiscuous removal of this plug, and since the live contacts introduce a possible ignition hazard when left unguarded, it was later required that arrangements be made to lock the plug in its receptacle. Incorporated in the design of the plug is a clamping arrangement intended to relieve the connections of any strains resulting from movements of the trailing cable. No cable reel is included in the equipment, therefore the cable must be taken care of by hand.

The chief features that distinguish the several types of machines from each other electrically are as follows:

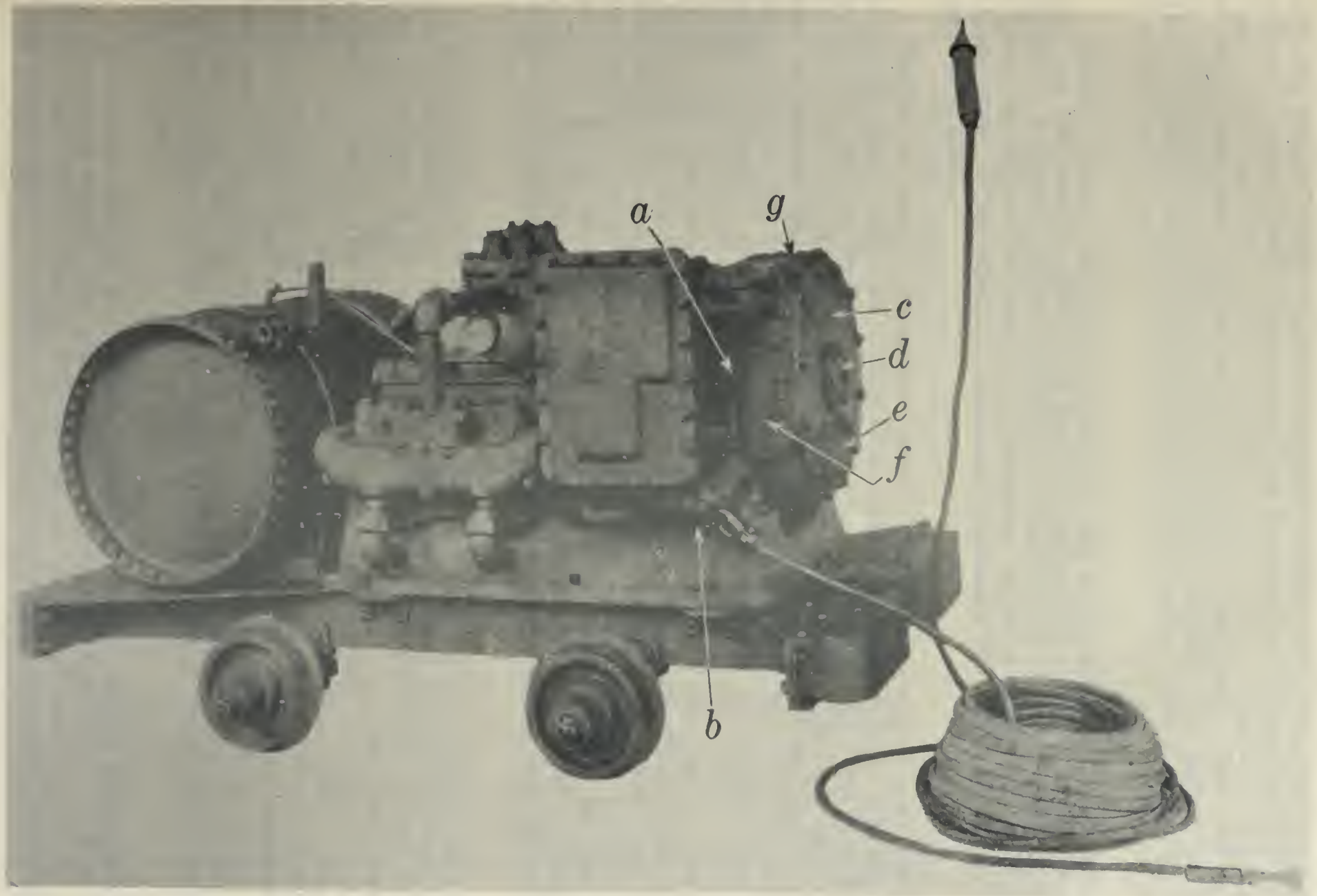


Figure 2.—General Electric type CP-26H compressor.

On the type CP-26D compressor the governor is housed in one enclosure while the master switch, thermal overload relays, and contactors are mounted in another. On type CP-26H compressors the governor is included in the same enclosure with the master switch and other parts, thus eliminating one enclosure. The CP-26G compressor corresponds to the CP-26H machine in that both have the governor in the same enclosure with the master switch. The former is built for 250-volt service and the CP-26H for 500-volt service. The type CP-26D machine is for 500 volts only.

Simple, rugged lines are followed in the construction of the enclosing cases for the various electrical parts of General Electric permissible air compressors. Explosion-proof qualities are secured for the most part by the use of heavy castings with broad, flat, machined surfaces in contact at the joints. A description of the essential features comprising the explosion-proof enclosures follows.

Except for rounded edges and corners the motor is almost cube shaped. One end of the motor frame has a relatively large circular opening through which the armature may be inserted or withdrawn. Bearing housing or framehead c that fits over this opening is held in place by 12 cap screws wired together in pairs to prevent their loosening. The framehead makes a "step joint" with the frame; that is, the machining presents a 90° change in direction to any flames tending to escape. This is accomplished by means of a lip on the head fitting closely within the circular opening of the frame, together with a surface in contact with the end of the frame and parallel to it. The bearing cap d is a circular plate that closes the opening into the bearing. This plate is held in place by six cap screws that are kept from loosening by wire threaded through holes drilled through their heads. Flat, machined surfaces in contact make a flametight joint between the motor frame and the bearing cap. The bearing itself is of the sleeve type. Openings in the inner parts of the housing for oil drainage necessitate use of the bearing cap to complete the explosion-proof enclosure.

To give access to the commutator and the four sets of brushes and brush holders, the motor has four openings with plate covers. One of those openings is at the bottom of the framehead. This opening is protected in the assembled motor by cover e. The explosion-proof joint between the cover and framehead consists of broad, flat machine surfaces held in contact by means of nine cap screws. Two other openings are on either side of the motor frame. The cover plate f, held by 10 cap screws, closes one of them. The fourth opening is on top of the motor frame and is of the same size as the two just mentioned. Cover plate g protects this opening. It is a duplicate of cover f and is fastened in the same manner. All of these plates depend upon close contact with flat surfaces on the motor frame to keep possible flames from escpaing from the motor.

The second bearing for the armature shaft is in reality an outboard bearing, since the support for the shaft is external to the motor. A cast sleeve closely resembling a bearing fits around the shaft and completes the closure of the pinion end of the motor. Although there is an allowable maximum radial

clearance of 0.006 inch between the shaft and sleeve, the length of this passageway is sufficient to keep flame from escaping.

Electrical connections to the motor consist of two single-conductor cables, each in a separate stuffing box. These stuffing boxes at first depended upon the insulation of the cables themselves to protect them against grounding on the magnet frame. Later, however, insulating bushings were put in as an additional precaution against grounding.

There is a small difference between the 250- and 500-volt motors, in that the framehead for 250-volt motors is made slightly deeper than that for 500-volt motors. In other words, the enclosed volume of 250-volt motors is made a little greater than for the 500-volt motors.

As has been indicated previously, the contents of the master switch box varies with the type of compressor. The box for the CP-26H compressor when first approved had in it the governor, a single-pole main-line contactor, a temperature relay, and a single-pole control switch. The box for the CP-26D compressor has two contactors, a temperature relay, and a double-pole control switch, the governor being in a separate explosion-proof compartment. The box for the CP-26G compressor, like that of the CP-26H compressor, contains a governor, a contactor, a temperature relay, and a single-pole control switch. The foregoing equipment did not provide for short-circuit protection, which was required by a revision of the schedule made subsequent to the approval of the machines. To comply with this later requirement, a single fuse was placed in the master switch box for both the CP-26G and H compressors. The general dimensions of the box itself are practically the same in each case, but the location and number of openings for cables and switch shafts are varied as required. Thus, for the CP-26D compressor there are six stuffing boxes for as many single-conductor cables that pass through the walls, besides two other openings to take the switch-operating shaft and reset rod, respectively. The box for the CP-26G and H compressors has four stuffing boxes for leads, one for the pipe connecting the governor with the air receiver, and the two openings for the reset rod and switch-operating shaft. However, these two are reversed in position in the two boxes.

Both boxes are made of cast iron, as are also the covers. The boxes are ribbed externally. The cover for the box on the CP-26H and G compressors, as shown in figure 2, has two adjoining, rectangular, shallow raised places to secure the desired electrical clearance for parts of the governor and contactor. The cover for the box on the type CP-26D compressor has just one raised place, but it is larger in area and includes most of the upper half of the cover, giving clearance for the two contactors mounted side by side in the upper part of the box. The cover for each box is held by 20 bolts with nuts secured by lock washers. The joint between the box and cover in either case consists of flat, machined surfaces in contact.

The openings in the boxes for the shafts and reset rods are protected against the discharge of flame by the small clearance of these parts in their holes as well as by the length of the holes.

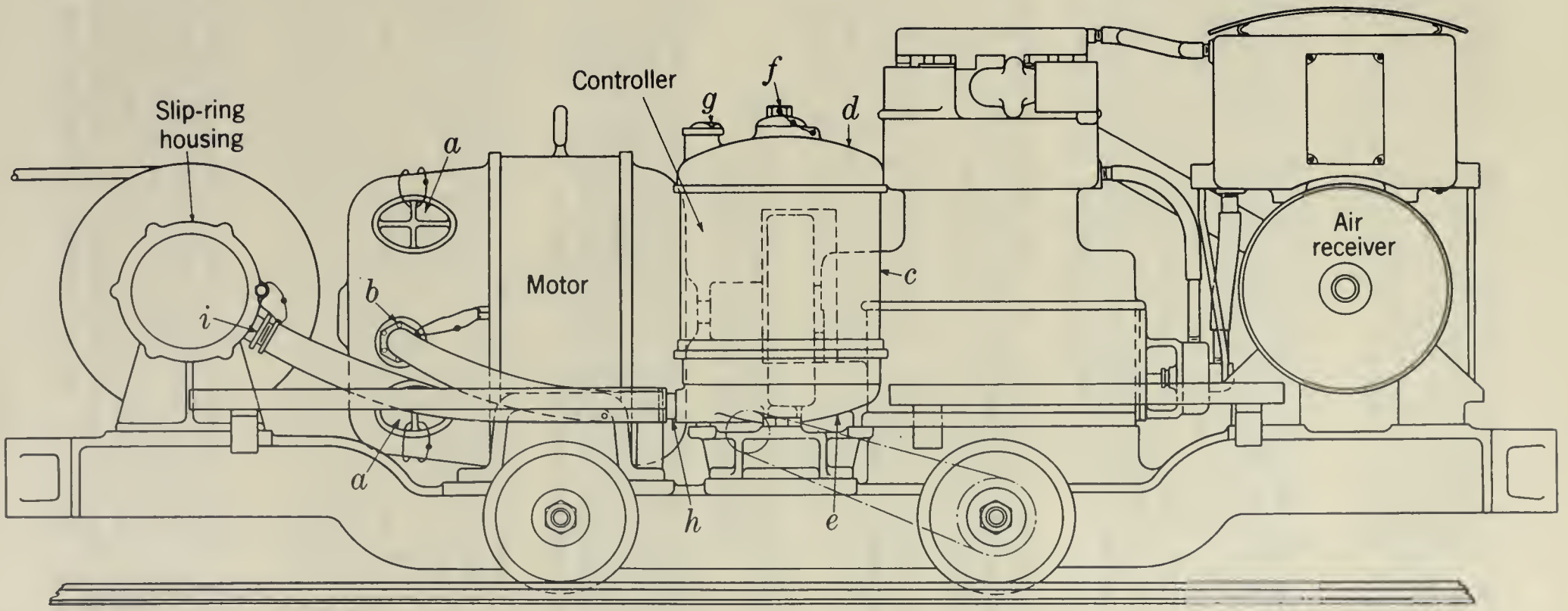


Figure 3.—Ingersoll-Rand type 20 compressor.

The governor box, which, as already stated, is used only on the type CP-26D compressor, resembles somewhat the master switch box in general appearance but is smaller. It is made of cast iron and is strengthened by external ribs. The cover also is of cast iron. The joint between the cover and box consists of wide, machined surfaces meeting in a plane and held together by 12 bolts with nuts secured by lock washers.

The governor box has seven openings through its walls. Four of these were drilled and tapped for studs used to hold the governor in place. The external ends of the four holes are closed with pipe plugs, the protruding ends of which are cut off after the plugs have been screwed into place. The other three openings are fitted with stuffing boxes. Two of these have a single-conductor cable in them, and the third admits an air pipe connecting with the governor. The two cables are a part of the control circuit and are connected to contacts that are opened and closed as the governor operates.

The stuffing-box gland nuts on each of the enclosures were made to clamp or hold the end of a piece of conduit and thus all wires and cables external to the enclosures, except the trailing cable, are encased in rigid iron conduits.

The trailing cable specified for all three types of compressors for either voltage is 250 feet of No. 4 parallel, duplex, all-rubber, insulated cable. This has a fused trolley tap and a rail clamp by which connection can be made to the trolley circuit for operating the compressor.

Ingersoll-Rand Co. Type 20 Compressor
Approvals Nos. 159 and 159A

On September 4, 1928, Approvals Nos. 159 and 159A were issued to the Ingersoll-Rand Co. covering their type 20, size 5-1/2 by 5 duplex compressor without cable reel for 250 and 500 volts, respectively. The use of a cable reel on compressors for either voltage was authorized by extension of approvals under date of October 17, 1928, and on May 11, 1931, the low-head compressor was authorized.

The approvals granted the Ingersoll-Rand Co. cover a duplex type 20 compressor in two different heights, one being approximately 45 inches over-all and the other about 55 inches in height. The higher machine is self-propelling and may be had either with or without a cable reel. Figure 3 shows this machine with a cable reel. The low-head machine is not self-propelling and does not have a cable reel. Power for driving the self-propelling unit is derived from an air motor that is connected to one axle by a chain and sprockets. The cable reel also is driven by an air motor that keeps tension on the cable regardless of the direction of travel.

The same design of motor and controller is used on both machines. The controller is operated by pushbuttons and arranged to give three-point time-limit acceleration with no-voltage protection. In other words, magnetic contactors close the circuits and cut out steps of resistance to start the motor. With failure of power supply it becomes necessary to press the "start" button again. Overload and short-circuit protection for the motor are afforded by an overload

relay that opens the main circuit in case either of these contingencies arise. A trailing cable terminating in a fused trolley tap and rail clamp is connected to the trolley or other power circuit in the mine for the energy necessary to operate the compressor.

The motor and controller used on these compressors are built by the Westinghouse Electric & Manufacturing Co. Some of the electrical parts for the cable reel also are furnished by this manufacturer, but the explosion-proof enclosure for the electrical parts is built by the Ingersoll-Rand Co.

Three distinct types of joints are used in the construction of the housings for the electrical parts of the Ingersoll-Rand permissible compressors. The main joints of the motor are of the "step" type, those of the controller are cylindrical, and the joints of the cable reel slip-ring housing are made up of flat, machined surfaces.

The motor, which has a continuous rating of 15-horsepower, is designated as a 113-SK motor. The frame is made up of a piece of steel rolled to form a cylinder, after which the ends are welded together to form one continuous magnetic circuit. Steel forgings riveted to the frame constitute feet for the motor. The two bearing brackets are of cast iron. The front bracket encloses the commutator of the motor. It has four openings equally spaced, through which adjustment of the brushes can be made and the commutator inspected. These openings are threaded to take brass covers, a, figure 3. During normal operation of the compressor these covers are secured with a seal to prevent unauthorized opening.

Each bracket is made with a lip that fits closely in the frame. Both also have a shoulder fitting against the end of the frame. Thus a section through the joint between the bracket and frame shows a 90° change in direction in the path from the interior to the exterior of the motor. Eight cap screws secured with lock washers hold each bracket to the frame.

In addition to the four inspection openings in the front bracket there is also a tapped hole for a stuffing box at b, figure 3, through which a three-conductor cable enters the motor. Each bracket has an opening into which a ball bearing fits. Two parts are necessary to complete each bearing housing. The inner housing cover or cap at each end has four studs screwed in it and secured by welding. These extend through holes in the bracket and are long enough to pass through the outer bearing cap, which is then held by nuts secured with lock washers. Each cap makes a step joint with the bracket; that is, it is machined so that one portion fits inside the opening for the bearing and another portion at right angles to it seats against the bracket. The outer cap at the commutator end is made solid because the armature shaft is not extended through at this end. The inner cap at this end and those at the other end fit the shaft closely enough so that, in conjunction with the bearings themselves, the passage of flame from explosion within the motor is stopped effectively.

There are 12 holes through the motor frame that are filled with bolts holding four main field poles and two interpoles. Another hole in the top of the frame is filled with an eyebolt permanently secured by welding.

There are three principal parts to the enclosure for the controllers - the case, the cover, and the base. Case c is made from a piece of wrought-iron pipe machined at the ends. The cover d and base e are steel castings that slip over the ends of the case. The three parts are held together by a tie rod of cold-rolled steel 1-1/2 inches in diameter. This rod is threaded at both ends and screws into a tapped hole through the base, where it is prevented by a key from loosening. The other end of the rod comes almost flush with the top of the cover, and the opening in the cover is made somewhat larger than the rod to receive the body of a special brass nut, f, which is screwed on the rod to clamp the parts of the enclosure tightly together. This nut is secured with a seal.

The rod just mentioned serves a double purpose. In addition to its function as a tie rod to hold the parts of the enclosure together, it also supports the panel on which are mounted the starting resistor, contactors, relays, and other parts necessary to the proper control and protection of the motor.

The main joints in the controller enclosure are made up of machined cylindrical surfaces at the ends of the case fitting inside finished surfaces of the cover and base. The narrow rim at the ends of the casing seats against a shoulder in both the latter. The length and fit of these joints are such as to stop the escape of flames following an explosion of a methane-air mixture within. Likewise, the joint between the special nut and the cover is made flametight by long, closely fitting, cylindrical surfaces in addition to a shoulder on the nut seating against the top of the cover. Two other holes pierce the cover. These are filled with close fitting plunger rods that operate the pushbuttons. A head turned on the inner end of each plunger prevents it from coming out and also interposes another directional change in the path of possible flames, thus effectively stopping their escape. The external ends of the plungers are in a pocket on top of the cover and are protected by a small hinged cover, g.

Besides the opening in the base for the tie rod there are two other openings at h. Each of these contains a stuffing box, one of which takes a three-conductor cable connecting the motor and controller while the other takes a two-conductor cable. The latter may be a short piece of cable between the reel and controller, as indicated in figure 4; or else, in case a cable reel is not included, it may be the trailing cable. If it is the trailing cable, a bell-mouth fitting is used at the cable entrance instead of a fitting for holding hose conduit.

The slip-ring enclosure for the cable reel has two principal parts, a housing and cover, both of which are made of cast iron. The general shape of the assembled enclosure is a short cylinder split at right angles to its axis. The two parts meet in a plane with machined surfaces wide enough and held tightly enough together to retain flames. Six bolts are used to hold the two parts together.

A ball bearing is set in an external recess of the housing concentric with the axis of the latter. The inner race of this bearing fits on a hollow spindle that extends into the interior of the enclosure. The two slip rings are mounted on the internal end of the spindle, properly insulated from it. The other end of the spindle is held by a press fit in the drum on which the cable is wound. Part of the drum extends into the same recess that holds the ball bearing. This extension rotates within the housing, which remains stationary; the clearance is small. This, together with the obstruction formed by the ball bearing itself, suffices to retain flames when an explosion takes place in the enclosure. There are two other holes in the housing, each containing a brush-holder rod welded in place. The rods support rigging in which are mounted the brushes, which ride on the collector or slip rings. The hollow spindle has an asbestos-packed stuffing box at its outer end through which the trailing cable enters and connects with the collector rings. A clamp in the drum eliminates strain upon the cable in the stuffing box.

The electrical connections between the slip-ring enclosure and the controller are made by means of a two-conductor cable that enters both enclosures through asbestos-packed stuffing boxes. The stuffing box at the cable reel enters the slip-ring housing radially.

The stuffing-box gland nuts on all three enclosures are secured against loosening by means of seals, and all but the one in the hollow spindle have extensions that fit inside hose conduits that give mechanical protection to the cables. When there is no cable reel the only hose conduit is that between the motor and controller. Then, as already stated, the trailing cable enters the controller by way of a bell-mouth gland nut, and a clamp fastened to the frame or chassis of the compressor holds the cable so that strains will not be transmitted to the stuffing box or to the connections inside. This clamp supplements one in the bell-mouth entrance. On the low-head compressor it was found necessary to add a support for the conduit between the motor and controller.

The trailing cable authorized for use with these compressors is of the all-rubber, insulated type in sizes 2, 3, or 4, the length being that necessary to suit the customer's requirements. A fused trolley tap and rail clamp are the means specified for attaching the cable to the power supply.

Sullivan Machinery Co. Type WK-22 Compressor
Approvals Nos. 160 and 160A

A third type of compressor built by the Sullivan Machinery Co. was known as its type WK-22 machine. It was approved on September 10, 1928, for both 250- and 500-volt direct-current service. The 250-volt machine is identified by Approval No. 160 and the 500-volt machine by No. 160A. This type was somewhat lower in over-all height than those previously considered. The machine differs in several respects from those already considered. Instead of one large-diameter air receiver two of small diameter are used. Another departure from the conventional design of the previous machines is the employment of a

water pump and radiator for circulating and cooling the water for the jacket of the compressor. Furthermore, the control is pushbutton instead of manually operated.

Briefly, the operation and functions of the controller unit are as follows: When the "start" button is pressed a control circuit is energized, causing certain contactors to close in proper time sequence and gradually bringing the motor up to speed as steps of resistance are cut out by the contactors. Time relays govern the interval between the closing of successive contactors. An overload relay is so arranged that an overload will cause it to open the control circuit, after which the contactors will drop open and shut down the motor. The control circuit will then remain open until the "stop" button is pressed or the main-line switch mounted on the controller is opened. In other words, after the motor has been stopped by an overload it can be started again by pressing first the stop and then the start button or else by opening and closing the main-line switch. If the motor is stopped merely by failure of the supply voltage, the relays, etc., are so adjusted that they will return to their normal positions for starting, and the motor will then automatically start again when the supply voltage is restored.

In addition to the overload relay, the equipment is protected further by a fuse in the plug of a main-line switch, which is mounted on the cover of the control unit.

Except for some minor details, such as a shaft extension through the front head for driving the water pump, the motor is essentially the same as that used on the two Sullivan compressors already described.

No cable reel is provided on the WK-22 compressor, and consequently the trailing cable requires coiling and uncoiling by hand from available space on the machine.

There are three distinct explosion-proof enclosures on the type WK-22 compressor - the motor, the control unit, and the main-line switch. Broad machined surfaces in contact at the joints of these three enclosures serve as the principal means of protection against discharge of flame from chance explosions of mine gas within them. A brief description of the essential constructional features follows:

The motor is designated as the type CY-24 and has a rating of 30 horsepower for 1 hour. The enclosure consists of three main parts - the magnet frame, the front head, and the rear head. The magnet frame, as used with the motor for this compressor, does not differ from that on the type WK-39 compressor and, aside from a different bearing cap with a hole drilled through it for the shaft extension that drives the water pump, the front head also is the same as that on the WK-39 compressor. The rear head, however, was changed in several respects to adapt it to the WK-22 compressor. Formerly, the controller was fastened to the side of the motor. This necessitated the use of holes in the rear head for the studs to hold the controller. It also required other holes in this head for insulated studs making electrical connection between the two.

These two sets of holes were eliminated from the rear head on the WK-22 compressor. The lead entrance was changed from six insulated studs to a single stuffing box for a three-conductor cable. With the change in type of lead entrance there was also a change made in its position.

The exceptions just noted affected the possible flame paths to a certain degree, but, as already indicated, the main joints in the motor remained unchanged. For details of their construction reference should be made to the motor described under the Sullivan type WK-26 compressor.

The control panel and starting rheostat are housed in a rectangular cast-iron box with a shallow cast-iron cover held in place by 22 studs with nuts and lock washers. The joint between the cover and box consists of broad, flat, machined surfaces in contact with each other. Midway between the two ends of the cover the outside surface has been planed to take the main-line switch box. There are four drilled holes through the cover for insulating bushings that carry the leads between the switch and the controller. The leads fit tightly enough in the bushings to prevent the propagation of flame from one compartment to the other. In addition to the four holes just mentioned, there are three others in the controller cover. Two of these are reamed to provide a close fit 1-1/4 inches long for the pushbutton rods, and the assembly effectively obstructs the passage of flame. The other hole is just beneath the pushbuttons and takes a stuffing box through which the trailing cable enters. In the back wall of the controller box is one more hole. It is fitted with another stuffing box, through which passes a 3-conductor cable making connection between the controller and the motor. This cable is encased in a length of rubber air-hose conduit for protection against mechanical injury.

The main-line switch used on this compressor is the same as that used on the other two Sullivan machines previously described; that is, it contains a two-pole, single-throw, knife-blade switch interlocked with a fuse plug. Details of the construction are given under the description of the type WK-26 compressor.

The trailing cable specified in the approval of the type WK-22 compressor was 250 feet of No. 4 all-rubber insulated cable terminating in a rail clamp and fused trolley tap. The stuffing-box entrance was fitted with a bell-mouth cover, and an insulated clamp grips the cable and prevents strains from being transmitted to the connections inside the control box. In addition, an insulated clamp held by a chain and hook is provided to hold the cable. It is essential that the cable be connected properly with respect to polarity, as both the fuse and overload relay are in the positive side of the circuit.

Sullivan Machinery Co. Redesigned Type WK-22 Compressor Approval No. 189.

The Sullivan Machinery Co. redesigned type WK-22 air compressor was the fourth made by that manufacturer to receive Bureau of Mines approval. It was approved on April 22, 1930. This approval covers only 250-volt machines. The approval number assigned was No. 189.

The redesign consists in the use of a Westinghouse motor and control. In this design a pushbutton-operated controller is again used, and a cooling system with radiator and pump likewise is retained. The over-all dimensions of the new compressor remain virtually the same as for the original type WK-22 machine.

The controller is automatic in operation. One pushbutton governs the starting and stopping of the motor, and the second is used to reset the overload relay when that relay functions. When the "start" button is pressed the main-line contactor in the positive line to the motor is closed. Time-limit relays then operate and cause the main contactor in the negative line to close, thus placing the motor across the line with starting resistance in series with the armature but with full voltage impressed on the shunt field. Following this, two other contactors close in proper sequence and time to short-circuit the starting resistance and bring the motor up to full speed. To stop the motor the start button must be released. In case of voltage failure while the compressor is running the relays resume their starting positions, and when the voltage is restored the starting will take care of itself. The overload relay in this controller gives the necessary overload and short-circuit protection to the motor.

No cable reel is included in the equipment on the redesigned type WK-22 compressor. Therefore, the trailing cable, when not in use, must be placed on the machine by hand in whatever space is available, and when it is to be used it must be removed by hand.

The construction used in securing explosion-proof enclosures for the motor and controller followed comparatively simple lines. Both enclosures were made readily accessible.

The motor is of the SK type built by the Westinghouse Electric & Manufacturing Co., and the frame is designated by the number 111-L. It has compound winding and a continuous rating of 20-horsepower. The construction of this motor is essentially the same as that of the 113-SK motor described under the Ingersoll-Rand compressor. The main point of difference between the two lies in the fact that the 111-L frame is 7-3/8 inches longer than the frame of the 113-SK motor. As was explained in that description, the joint between the bearing brackets and the frame is of the step type. The bracket at the commutator end also has four inspection openings with screw-type covers.

The enclosure or casing that houses the control mechanism also houses the starting rheostat. The elevation of this enclosure is oblong in general shape, with the longer axis horizontal. The casing has two main parts - a box made of semisteel and a cover of the same material. The thickness of metal in the cross-sections of these two parts is not great, but both parts are strengthened by internal and external ribs closely spaced. The joint between the two in their normal position when mounted is vertical. The joint itself is of the tongue-and-groove type, having a tongue machined on the cover that fits into a groove machined in the box. The fastenings for holding the cover to the box are six tap bolts with lock washers. Although this number is small in relation to the perimeter of the joint, the size of the bolts (1-1/4 inches) and the stiffness of the joint, together with its special form, are adoquate to prevent the escape of flame should an explosion of methane and air take place within the enclosure.

Two openings in the right side of the box are each fitted with a stuffing box. The upper one admits the three-conductor cable that connects the motor with the controller. This cable is protected by hose conduit, which is secured at each end to an extension on the gland nut. The lower opening is the entrance for a two-conductor rubber-sheathed trailing cable. This opening has a bell-mouth fitting, which holds a split clamp of insulating material. The purpose of the clamp is to keep any strains on the cable from being transmitted to the connections within the control casing, and the bell-mouth restricts the degree of bending in the cable at the entrance.

The controller cover has two 3/8-inch-diameter holes drilled through it for plungers that operate the pushbuttons. These holes are 1-1/16 inches long. The plungers are 3/8-inch in diameter in the shank but have a head of larger diameter on the inner end, which prevents the plunger from being blown out should an explosion occur inside of the controller. Normally the shoulder between the head and shank is kept seated against a machined face by a spring. This fact, together with the length of the hole and the close fit of the plunger in it, makes the assembly flametight.

A No. 3 all-rubber insulated cable of suitable length (between 250 and 500 feet) was specified as the trailing cable to be used with the approved type WK-22 compressor. This cable terminates in a rail clamp and fused trolley tap for connection to the trolley or other convenient power circuit. It is essential to have the trailing cable correctly connected with respect to polarity so that the overload relay is in the positive side of the circuit.

CONCLUSION

In the six compressors just described, the prospective purchaser will find equipment that has successfully passed rigid inspections, explosion tests, and "adequacy" tests through which any inherent weakness in design and construction is disclosed. The inspections give visual evidence of failure to comply with the requirements specified in the schedule. The explosion tests demonstrate the ability of motor, controller, and other enclosures to withstand explosion pressures within them without releasing flames through any joints. By means of adequacy tests inadequate electrical clearances and other defects that might lead to hazards of shock or of burning holes through casings intended to be explosion-proof, are discovered. Thus, the mine safety engineer is assured that the permissible air compressor is to be preferred over any unapproved type when the question of safety is involved.

Each permissible compressor is identified by an "approval plate", which carries the official seal of the Bureau of Mines, the approval number of the machine, and the name of its manufacturer. This plate also includes general instructions to be followed in order to maintain the compressor in a permissible condition.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

622.09
R.I. 3310
Un32mo
Cop2

APRIL 1937

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

EXTRACTION METHODS FOR DETERMINING TAR ACIDS AND BASES, AND VARIABLES AFFECTING THEIR ACCURACY

BY

C. H. FISHER AND ABNER EISNER

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R. I. 3310
April 1937

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

EXTRACTION METHODS FOR DETERMINING TAR ACIDS AND BASES, AND VARIABLES AFFECTING THEIR ACCURACY[1].

By C. H. Fisher[2] and Abner Eisner[3]

CONTENTS

	Page
Introduction	2
Acknowledgment	3
Materials and procedure	3
Important variables and possible errors in extraction with alkali	3
Alkali concentration	3
Molecular weight of amine in toluene solution	5
Sodium phenolate concentration	7
Other variables and possible errors in extraction with alkali.	10
Important variables and possible errors in extraction with acid	10
Acid reagent	10
Concentration of amine in acid solution	12
Molecular weight of amine in acid solution	13
Other variables and possible errors in extraction with acid	14
Analytical procedures for tar acids and bases	14
Bureau of mines method	14
Comparison of several extraction procedures	17
Analysis of oils containing carboxylic acids	20
Correction for water absorption	22
Procedure recommended for determination of tar acids and bases	24
Summary and conclusions	25

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3310".

2/ Associate organic chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines, Pittsburgh, Pa.

3/ Junior organic chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines, Pittsburgh, Pa.

Page

Appendix A. Important variables and possible errors in extraction with alkali 27
Concentration of amine in toluene solution 27
Hydrocarbon solvent 28
Molecular weight of sodium phenolate 28
Appendix B. 30
Distribution of aniline between cresol-toluene and aqueous acid solutions 30
Compound formation between tar acids and bases 31

INTRODUCTION

Many methods of analyzing tar oils for their tar acid and base content have been employed in connection with both academic and industrial investigations. Most of these analytical procedures are based on the diminution in volume caused by extraction with alkali and mineral acids (contraction method), although other methods have been employed[4]. In many instances the adoption of an analytical method was not preceded by adequate study, and, as a consequence, the literature on this subject is conspicuous for its contradictory claims and recommendations. It has been shown in the case of certain synthetic solutions[5] that large errors may result if the analysis is carried out by simple extraction with alkali and mineral acids.

It was the purpose of the present work to study and compare several modifications of the contraction method and to show under what general conditions tar acids and bases can be extracted accurately. Synthetic solutions, coal-tar oils, and hydrogenated coal distillates were used in this connection, and methods of determining tar acids as phenols and carboxylic acids were investigated. Methods that include washing the alkaline and acid extracts with organic solvents were not studied, since such methods are wasteful of time and subject to errors of prolonged manipulation. Furthermore, washing the aqueous extracts with organic solvents fails to accomplish the desired end, since, for example, an organic solvent removes phenols as well as non-acidic materials from alkaline solutions.

4/ Heinze, R., and Marder, M., The Testing of Tar Oils. Determination of Creosote Content: Chem. Trade Jour. 1936, p. 85; Lugovkin, B. P., Volumetric Method of Determining Phenols by Means of Glycerol in Distillation Tars of Brown Coals: Jour. Chem. Ind. (Moscow), vol. 5, 1928, pp. 798-799 (Chem. Abs., vol. 23, 1929, p. 957); McKee, R. H., and Parker, H. H., Determination of Nitrogen Bases in Petroleum Oils: Ind. Eng. Chem., vol. 19, December 1927, pp. 1343-4; Ubaldini, I., and Ciarrocchi, I., New Method of Determining Phenols in Tar: Acqua e gas, vol. 21, 1932, pp. 124-127 (Chem. Abs., vol. 27, 1933, p. 4058); von Walther, R., and Lachman, K., Titration of Brown-Coal-Tar Creosote with Naphthalendiazonium Chloride: Braunkohlen-archiv, No. 26, 1930, pp. 1-11 (Chem. Abs., vol. 25, 1931, p. 3465); and Morgan, J. J., and Meighan, M. H., Improved Methods for Determination of Phenols in Tar Oils: Ind. Eng. Chem., vol. 17, 1925, pp. 626-628.

5/ Kester, E. B., and Rockenbach, L. P., Unpublished work, U. S. Bureau of Mines, Pittsburgh Experiment Station.

It has been realized for some time that the chief errors of extraction methods arise from the fact that nonacidic material is removed with the tar acids in the alkaline extraction, and, conversely, nonbasic material with the tar bases in the mineral acid extraction. Accordingly, in the present work special attention was directed to the magnitude of these undesirable side extractions and the variables that control them.

ACKNOWLEDGMENT

The authors are indebted to J. D. Davis and L. P. Rochenbach for supplying coal tars and to Dr. H. H. Storch and Dr. T. E. Warren for providing samples of hydrogenated coal. Special thanks are due to Doctor Storch for his kind examination of the manuscript and helpful suggestions.

MATERIALS AND PROCEDURE

The synthetic solutions studied were prepared from good-grade chemicals with pure toluene as the hydrocarbon solvent. The cresol ("tricresol") and xylidine were mixtures of isomers, and the pseudo-cumidine was of technical grade.

The extractions were carried out in funnels[6] graduated to 100 cc in 0.2-cc subdivisions. About 1 hour was allowed to elapse between agitation of the oil (100-cc samples) with the desired aqueous solution and observation of volume changes.

IMPORTANT VARIABLES AND POSSIBLE ERRORS IN EXTRACTION WITH ALKALI

Alkali Concentration

There is considerable disagreement concerning the concentration of alkali best suited for the accurate extraction of tar acids, and a number of alkaline reagents have been recommended. It appears certain that "maximum extractions" are obtained with rather concentrated alkali[7]. However, it is generally recognized that the alkaline extracts thus obtained contain considerable quantities of nonacidic material, due to the solvent power[8] of alkaline-phenols solutions.

6/ Weiss, J. M., Methods of Analysis Used in the Coal-Tar Industry. III. Heavy and Middle Oils: Ind. Eng. Chem., vol. 10, November 1918, pp. 911-916; Gas Chemists Handbook, 3d ed., 1929, p. 433, Tar-Acid Separatory Funnel, Barrett Type No. 2.

7/ Cheng, Y. C., and Morgan, J. J., The Extraction of Tar Acids from Low-Temperature Coal Tars: Gas Age-Record, vol. 59, June 1927, pp. 815-816, 818; Morgan, J. J., and Meighan, M. H., Extraction of Phenols from Tar Oils by the Caustic Soda Process: Ind. Eng. Chem., vol. 17, 1925, pp. 696-700; Rachovskiĭ, V. E., and Ioffe, P. M., The Phenolate Method of Extracting Phenols from Acidic Tar Oils: Khim. Tverdogo Topliva, vol. 3, 1932, pp. 733-743 (Chem. Abs., vol. 28, 1934, p. 5212).

8/ Schoutle, R., The Miscibility of Solutions of Phenols in Aqueous Alkalies with Organic Substances Insoluble in Water: Ann., vol. 351, 1906, pp. 473-480.

In many instances nonacidic material has been removed from the alkaline layer by washing with organic solvents. Probably 10 percent sodium hydroxide[9, 10] is employed most frequently as the alkaline reagent, although 5[11], 15[12], 20[13], and 30 percent[14] have been used. It has been claimed[15] that the addition of sodium chloride to 10 percent sodium hydroxide tends to prevent the dissolution of nonphenolic or nonacidic substances.

In the present work the effect of alkali concentration was studied in one series of experiments by extracting a standard amines-toluene solution (100-cc portions) with 50-cc portions of sodium hydroxide solution.

Pyridine	12 cc	.....	1.85 percent
Isoquinoline	32 cc	.....	4.9 "
Aniline	56 cc	.....	8.6 "
Toluene	550 cc	.....	84.65 "

All the sodium hydroxide solutions examined (5, 10, 20, and 30 percent) failed to dissolve appreciable quantities of the amines-toluene solution, although all the amines present are fairly soluble in water. The potent

---

9/ Fieldner, A. C., and Davis, J. D., Gas-, Coke-, and Byproduct-Making Properties of American Coals and Their Determination: Monograph 5, Bureau of Mines, 1934, p. 145; Franta, W., and Lavine, I., Development of Dakota Lignite. XI: Ind. Eng. Chem., vol. 28, January 1936, pp. 119-121 (Gas Chemists Handbook, 3d ed., 1929, p. 431); Hill, J. B., Determination of Phenols in Coal-Tar Oils and Crude Carbolic Acids: Ind. Eng. Chem., vol. 15, 1923, pp. 799-800; Stadnikov, G. L., Methods for the Analysis of Solid Fuels: Khim Tverdogo Topliva, vol. 5, 1934, pp. 193-200 (Chem. Abs., vol. 28, 1934, p. 7467).

10/ Morgan, J. J., and Meighan, M. H., Work cited (footnote 4).

11/ Lazar, A., Gravimetric Estimation of Creosote: Gas Jour., vol. 153, 1921, p. 622; Morgan, G. T., Pratt, D. D., and Ross, J., A Chemical Study of Low-Temperature Tar: Jour. Soc. Chem. Ind., vol. 48, 1929, pp. 29-34T.

12/ Greenbaum, F., The Extraction of Phenol from Low-Temperature Tar: Chem. Ztg., vol. 26, 1923, 147-9 (Chem. Abs., vol. 18, 1935, p. 746) Greenbaum, F. R., The Extraction of Tar Acids from Low-Temperature Tar with Sodium Hydroxide: Catalyst, vol. 5, 1927, pp. 6-10 (Chem. Abs., vol. 21, 1927, p. 2975).

13/ Rakovskiĭ, V. E., and Ioffe, P. M., The Phenolate Method of Extracting Phenols from Acidic Tar Oils: Khim. Tverdogo Topliva, vol. 3, 1932, pp. 733-743 (Chem. Abs., vol. 28, 1934, p. 5212); A Study of the Tars and Oils Produced from Coal: Fuel Res. Tech. Paper No. 32, 1931, p. 45; Actions of Solvents on Coal: Fuel Res. Tech. Paper No. 37, 1933, p. 191.

14/ Pieters, H. A. J., Methods for Determination of the Phenol Content of Tar Oils: Het Gas, vol. 49, 1929, pp. 472-474 (Chem. Abs., vol. 24, 1930, p. 945).

15/ Morgan, G. T., Pratt, D. D., and Ross, J., Work cited (footnote 11).

influence of sodium hydroxide in decreasing the solubility of amines in water is shown well by the data in table 1. Although even 5-percent alkali dissolved only negligible quantities of the amines-toluene solution, washing with three 50-cc portions of distilled water caused a total diminution in volume of 2.5-3.0 cc.

The results in table 1 seem to indicate that, excluding factors other than the solubility of amines in aqueous alkali, almost any concentration of sodium hydroxide would be suitable for the analytical extraction of tar acids.

TABLE 1. - Extraction of amines-toluene solution with alkali

Extracted with 50 cc portions of-	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
5% NaOH	0.0	-0.1	-0.2	-	-0.3
	.0	- .3	- .1	-	- .4
10% NaOH	.0	.0	- .2	-0.2	- .2
	- .1	- .1	- .3	- .2	- .5
20% NaOH	+ .2	+ .1	- .1	- .1	+ .2
	+ .1	+ .1	.0	- .1	+ .2
30% NaOH	.0	.0	+ .1	- .1	+ .1
	+ .1	.0	- .1	.0	.0
20% NaOH saturated with NaCl	+ .1	.0	- .1	- .1	.0
	+ .1	.0	.0	- .1	+ .1
Water saturated with NaCl	+ .1	- .1	- .5	- .3	- .5
	+ .2	- .1	- .1	- .3	.0
Water	- .8	-1.2	-1.1	- .8	-3.1
	- .8	- .9	- .9	- .8	-2.6

Molecular Weight of Amine in Toluene Solution

Although aqueous alkali extracts only negligible quantities of amines (such as pyridine, aniline, etc.) from hydrocarbon solutions, it is well-known that alkaline solutions containing phenols have considerable solvent action for amines as well as other types or organic compounds16/. Among the amines of similar structure the molecular weight has a very noticeable effect

16/ Scheuble, R., Work Cited (footnote 8).

on the amount extracted by aqueous solutions of alkali and phenols, and amines of low molecular weight are removed more readily than those of high molecular weight (see table 2). In studying the effect of molecular weight, toluene solutions (100 cc) containing 25 percent by volume of amines of different molecular weight were washed successively with 50-cc portions of an alkaline-cresol solution (30 cc of cresol added to 100 cc of 20-percent sodium hydroxide). This concentration of cresol was used as a matter of convenience to produce large, and hence easily compared, volume changes in the amines-toluene layer. The toluene solutions containing solid amines were prepared by adding the amine to 150 cc of toluene until the total volume was 200 cc. Only the factor of molecular weight was considered, although other differences existed, such as basicity and molecular structure.

TABLE 2. - Solubility of various amines (as 25-percent solutions in toluene) in alkaline-cresol solution

Amine	Mol. wgt.	Volume changes of toluene layer, cc				
		1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
Pyridine	79	-12.4	-8.5	-5.2	-3.3	-26.1
Aniline	93	- 3.4	-3.2	-2.9	-2.6	- 9.5
		- 3.5	-3.5	-3.2	-2.9	-10.2
o-toluidine	107	- 1.0	-1.4	-1.3	-	- 3.7
		- 1.2	-1.6	-1.5	-	- 4.3
Xylidine	121	- 1.0	-1.4	-1.2	-1.2	- 3.6
		- 1.0	-1.3	-1.1	-1.3	- 3.4
Quinoline	129	- 4.7	-4.9	-4.3	-3.6	-13.9
Cumidine	135	- .9	- .9	- .8	-	- 2.6
		- .8	-1.0	- .9	-	- 2.7
α-naphthyl-amine	143	- .4	- .7	- .9	- .8	- 2.0
		- .3	- .6	- .8	- .8	- 1.7
Diphenyl-amine ..	169	+ .3	- .2	.0	- .1	+ .1
		+ .4	.0	+ .1	.0	+ .5
Triphenyl-amine .	245	.0	- .1	- .1	- .1	- .2

The amount extracted varied greatly from one amine to another, and the amines of lower molecular weight were removed from the toluene layer with remarkable effectiveness by the alkaline-cresol solution. The hydrocarbon solvent, toluene, also was extracted in appreciable quantities in some of the experiments. Washing the remaining portions of the pyridine-toluene and

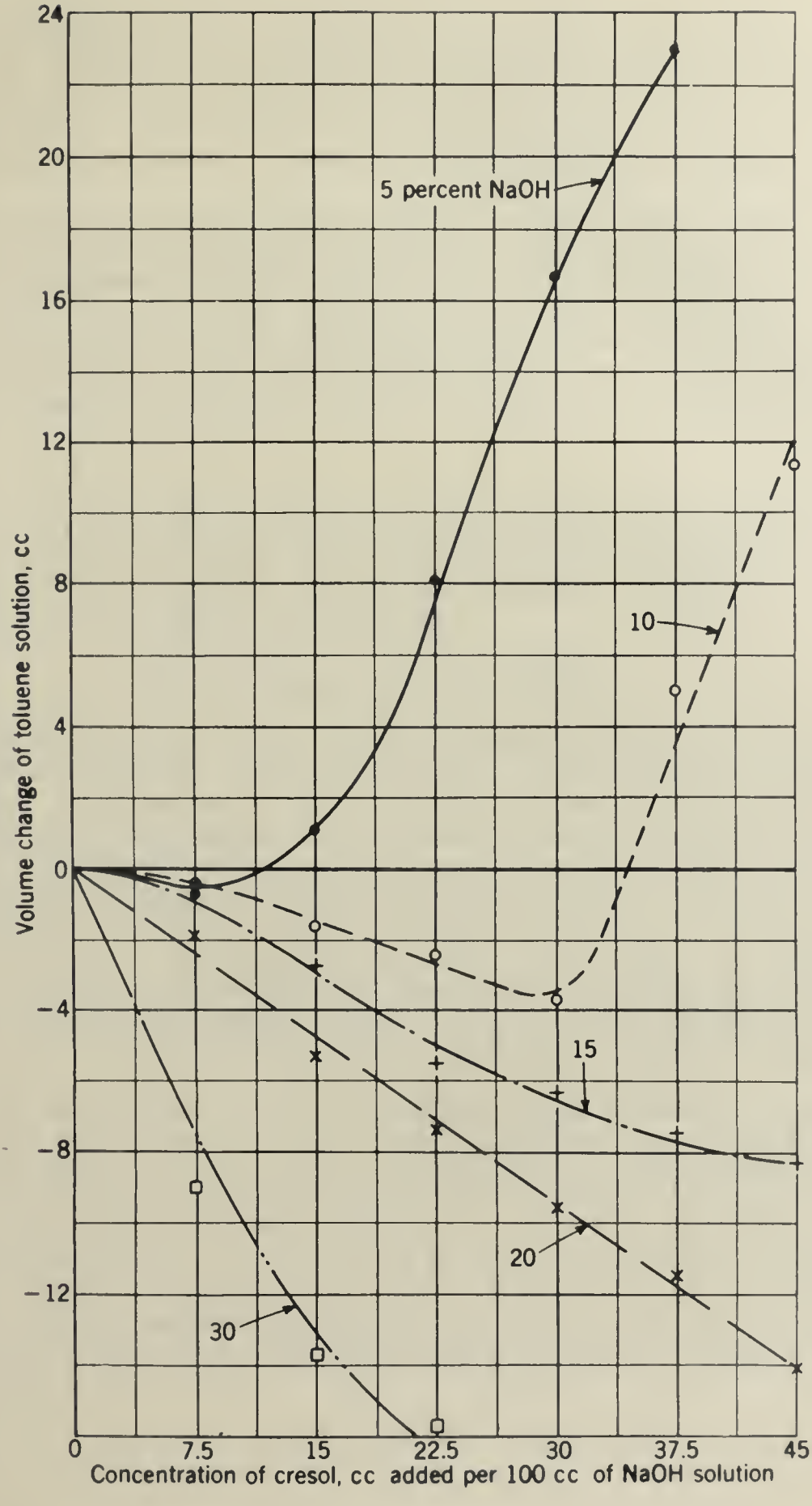


Figure 1.- Amines in toluene washed with 150 cc of alkaline-cresol solution.

quinoline-toluene layers with 20-percent hydrochloric acid caused further diminutions in volume of 6.9 and 12.6 cc, respectively, leaving 63.7 and 69.9 cc of toluene. This indicated that 11.3 and 5.1 cc of toluene had been removed from the pyridine-toluene and quinoline-toluene layers. Triphenylamine, as might be expected, behaved as a hydrocarbon or neutral compound.

On the basis of the data given above, it seems likely that amines are removed in considerable amounts by washing tar oils with alkali and that the amines thus extracted are chiefly those of low molecular weight. To prevent this undesirable removal of amines during the alkaline extraction, it would therefore be desirable in analytical procedures to remove the amines of low molecular weight prior to extraction with alkali. In a later section it will be shown that a preliminary wash with mineral acid removes chiefly the mines of low molecular weight.

The data for the second, third, and fourth extractions in table 2 (as well as in table 3) are generally little different from those obtained in the first extraction. These data are included for purposes of comparison with similar data (tables 4 and 5) where marked differences occur between the first and subsequent extractions.

Sodium Phenolate Concentration

In studying the effect of phenolate concentration on the solubility of amines in sodium phenolate solutions, the same amines-toluene solution used previously (p. 4) was washed several times in succession with 50-cc portions of aqueous solutions containing various amounts of sodium hydroxide and cresol. The alkaline-cresol solutions were prepared by adding certain quantities of cresol (7.5, 15, 22.5, 30, 37.5, and 45 cc to each 100 cc of sodium hydroxide) to several different concentrations (5, 10, 15, 20, and 30 percent) of alkali. The results (see table 3 and figure 1) indicate that the solubility of amines in alkaline-cresol solutions is directly proportional to the concentration of alkali and that large amounts of amines are extracted when the concentrations of alkali and cresol are high.

The effect of varying the cresol concentration is shown, also, by the data in table 3 and figure 1. It is readily apparent that the volume losses of the toluene solution were directly proportional to the concentration of cresol when sufficient alkali was present to neutralize the cresol. When the alkali present was not sufficient to react with the cresol the volume of the amines-toluene layer increased, due to the absorption of excess or "free" cresol. Because of the solvent power of concentrated alkaline-phenol solutions for amines, concentrated alkali should not be used to extract tar acids when, as in analytical procedures, the simultaneous removal of amines is not desired.

TABLE 3. - Extraction of amines-toluene solution with alkaline-cresol solutions of various concentrations

Concen. of NaOH, percent	Cresol added, cc per 100 cc	Volume changes of toluene layer, cc				
		1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
5	7.5	+0.1	-0.4	-0.3	-0.2	-0.6
		.0	- .4	- .4	- .3	- .8
10	7.5	+ .1	- .3	- .3	- .4	- .5
		+ .1	- .2	- .3	- .4	- .4
15	7.5	+ .1	- .3	- .2	-	- .4
		+ .1	- .4	- .2	-	- .5
20	7.5	- .4	- .7	- .9	-1.0	-2.0
		- .5	- .5	- .9	- .7	-1.9
30	7.5	-3.0	-3.3	-2.8	-	-9.1
		-2.8	-3.2	-2.9	-	-8.9
5	15	+ .9	+ .2	.0	- .1	+1.1
		+ .9	+ .1	.0	- .1	+1.0
10	15	- .3	- .7	- .7	- .6	-1.7
		- .3	- .6	- .7	- .6	-1.6
15	15	- .8	- .9	-1.0	-	-2.7
		- .8	-1.0	-1.1	-	-2.9
20	15	-2.0	-2.0	-1.6	-1.5	-5.6
		-1.9	-1.4	-1.7	-1.3	-5.0
30	15	-5.1	-5.2	-3.7	-2.7	-14.0
		-5.2	-4.9	-3.4	-3.3	-13.5
5	22.5	+3.1	+2.6	+2.4	-	+8.1
10	22.5	- .4	-1.0	-1.2	-1.0	-2.6
		- .3	-1.0	- .9	-1.1	-2.2
15	22.5	-1.8	-2.1	-1.7	-	-5.6
		-2.0	-1.9	-1.7	-	-5.6

(continued)

TABLE 3. - Extraction of amines-toluene solution with alkaline-cresol solutions of various concentrations (continued)

Concen. of NaOH, percent	Cresol added, cc per 100 cc	Volume changes of toluene layer, cc				
		1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
20	22.5	-2.6	-2.3	-2.1	-1.8	-7.0
		-2.8	-2.6	-2.3	-2.0	-7.7
30	22.5	-6.3	-5.1	-4.3	-2.8	-15.7
		-6.4	-5.3	-4.1	-3.0	-15.8
5	30	+5.5	+7.0	+4.3	-	+16.8
10	30	- .8	-1.4	-1.6	-1.3	- 3.8
		- .8	-1.4	-1.6	-1.3	- 3.8
15	30	-2.1	-2.3	-1.9	-	- 6.3
20	30	-3.7	-3.4	-2.5	-2.1	- 9.6
		-3.8	-3.1	-2.7	-2.3	- 9.6
5	37.5	+8.1	+7.4	+7.5	-	+23.0
10	37.5	+1.9	+1.7	+1.6	-	+5.2
		+2.0	+1.4	+1.6	-	+5.0
15	37.5	-2.8	-2.7	-2.2	-	-7.7
20	37.5	-4.7	-3.9	-2.8	-2.4	-11.4
		-4.8	-3.9	-3.0	-2.3	-11.7
10	45	+4.2	+4.3	+2.7	-	+11.2
		+4.4	+4.1	+3.0	-	+11.5
15	45	-2.8	-3.0	-2.5	-	-8.3
20	45	-5.9	-4.5	-3.3	-2.7	-13.7
		-6.4	-4.6	-3.4	-2.3	-14.4

Other Variables and Possible Errors in Extraction with Alkali

Data given in appendix A, table 13, indicate that in exceptional cases, when the tar oil contains more than about 5 percent of low-molecular-weight amines, extraction with alkali would result in relatively large errors, due to the solubility of tar bases in the aqueous layer. The magnitude of these errors can be reduced by extracting first with acid (see tables 6 and 8); in extreme cases it may be desirable to dilute the original sample with about an equal volume of toluene[17].

Appendix A also contains data on the effect of the nature of the solvent in which the amine occurs and of the molecular weight of the sodium phenolate in the aqueous layer.

IMPORTANT VARIABLES AND POSSIBLE ERRORS IN EXTRACTION WITH ACID

Acid Reagent

Dilute sulphuric or hydrochloric acid has been used to extract tar bases, inorganic salts being added in some cases to minimize the undesirable extraction of water-soluble tar acids.

A number of experiments were carried out to see if phenol, as a representative of the tar acids, could be extracted in appreciable quantities by dilute mineral acids. Approximately 100 cc of a phenol-toluene solution (80 g phenol in 500 cc of toluene) was washed in succession with four 50-cc portions of various aqueous solutions, and the acid solutions that caused little change in volume of the phenol-toluene layer were considered as possible acid reagents. The presence of inorganic salts, especially sodium chloride, was found, as anticipated, to retard the removal of phenol from the toluene layer. From the data in table 4 and the fact that sodium chloride would not be expected to depress the solubility of amine sulphates as much as inorganic sulphates, it appears that the most suitable reagent is 20-percent sulphuric acid containing sodium chloride.

The following experiment, which affords a more direct comparison of hydrochloric acid and the sulphuric acid-sodium chloride solution, has shown that the latter reagent is preferable. Washing 100 cc of a 15 percent by volume cresol-toluene solution with three 50-cc portions of a pyridine-hydrochloric acid solution (10 cc of pyridine added to 100 cc of 20-percent acid) caused the toluene layer to suffer a total volume loss of 2.6 cc. Under the same conditions the toluene layer gained 0.4 cc in volume when a pyridine-sulphuric acid solution (10 cc of amine added to 100 cc of 20-percent sulphuric acid saturated with sodium chloride) was used.

17/ See table 7.

TABLE 4. - Extraction of phenol-toluene solution with various aqueous solutions

	Extracted with 50 cc portions of-	Volume changes of toluene layer, cc				
		1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
1	20% H_2SO_4	+0.2	-0.8	-0.8	-0.7	-1.4
		+ .2	- .8	- .8	- .8	-1.4
2	20% H_2SO_4 saturated with cryst. K_2SO_4	+ .3	- .5	- .7	- .4	- .9
		+ .5	- .5	- .4	- .2	- .4
4	20% H_2SO_4 saturated with anhyd. Na_2SO_4	+ .6	- .2	- .3	- .1	+ .1
		+ .8	- .2	- .1	- .3	+ .5
5	20% H_2SO_4 saturated with NaCl	+ .3	- .1	- .3	- .0	- .1
		+ .5	- .3	- .3	- .5	- .1
6	20% H_2SO_4 saturated with NaCl and K_2SO_4	+ .7	- .1	- .2	- .3	+ .4
		+ .4	- .2	- .3	- .1	+ .1
7	20% HCl	- .3	- .9	- .9	- .7	-2.1
		- .6	- .9	- .8	-1.0	-2.3
8	20% HCl saturated with NaCl	- .2	- .8	- .4	- .6	-1.4
		.0	- .9	- .7	-1.0	-1.6
9	Water	- .4	-1.7			
		- .6	-1.4			
10	Water saturated with K_2SO_4	- .1	-1.3	-1.2	-1.3	-2.6
		- .2	-1.3	-1.1	-1.0	-2.6
11	Water saturated with NaCl	+ .4	- .4	- .3	- .4	- .3
		+ .5	- .2	- .2	- .6	+ .1

Concentration of Amine in Acid Solution

In order to obtain further information concerning possible errors due to the solubility of tar acids in the aqueous acid used to remove amines, a toluene-cresol solution (15 percent cresol by volume) was washed successively with several portions of 20-percent hydrochloric acid, to which had been added certain quantities of aniline for each 100 cc of acid. Hydrochloric acid was used in preference to sulphuric acid because of the greater solubility of amine hydrochlorides. As shown in table 5, the volume losses of the cresol-toluene solution were directly proportional to the aniline content of the hydrochloric acid solution. Whether this would be true for an acid solution containing an excess of aniline (that is, more than that neutralized by the acid present) was not determined.

Comparison of the first and second extractions of table 5 indicates that water probably passed into the cresol-toluene layer in appreciable quantities. The same phenomenon was observed less frequently and to a smaller degree in extracting amines-toluene solutions with alkaline-phenols solutions (tables 2 and 3). When water is absorbed by the oil layer during extraction, the volume diminution is not an accurate measure of the phenol or amine extracted, and hence water absorption is a source of error in analytical extractions.

TABLE 5. - Extraction of cresol-toluene solution with 20-percent HCl containing aniline in various concentrations

Aniline added to each 100 cc of 20 percent HCl	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
0 cc	0.0	-0.4	-0.2	-	-0.6
	.0	- .3	-	-	-
10 cc	+ .2	- .7	-	-	-
	+ .2	- .6	- .5	-0.6	- .9
20 cc	- .2	-1.1	-1.3	-1.1	-2.6
	- .1	-1.1	-1.2	-1.3	-2.4
30 cc	- .1	-2.2	-1.9	-1.7	-5.1
	- .7	-2.3	-2.2	-1.8	-5.2
40 cc	-4.1	-3.6	-1.9	-	-9.6
	-4.3	-3.7	-1.9	-	-9.9

Molecular Weight of Amine in Acid Solution

The solutions employed in studying the influence of the amine salt molecular weight were a cresol-toluene solution (15 percent by volume) and 20-percent hydrochloric acid, to which had been added 10 cc of the amine for each 100 cc of acid. Considerable quantities of cresol were removed by the acid solutions of amines of low molecular weight. As shown in table 6, large amounts of the amines of higher molecular weight[18] passed from the hydrochloric acid into the cresol-toluene layer. On the basis of the data in table 6, it seems likely that extraction of tar oils with mineral acid removes chiefly the low molecular weight amines, and that certain amines of higher molecular weight would be extracted only with difficulty. It is well known[19] that tar bases in general cannot be extracted conveniently in the presence of tar acids.

TABLE 6. - Extraction of cresol-toluene solution with 20-percent HCl (10 cc of amine added per 100 cc)

Amine hydrochloride	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
Pyridine	0.0	-1.0	-1.6	-	-2.6
	-	-	-	-	-
Aniline	+ .2	- .7	-	-	-
	+ .2	- .6	- .5	-0.6	- .9
o-toluidine.....	+1.7	-1.3	1/	1/	-
	+1.6	-1.4	1/	1/	-
Xylidine.....	+4.6	+3.7	-	-	-
	+4.8	+3.4	-	-	-
Quinoline.....	+1.6	- .6	-1.4	-	- .4
	+1.6	- .4	-1.4	-	- .2
Pseudo-cumidine.	+4.9	+5.4	+4.8	+2.6	+15.1
	+5.0	+5.4	+4.5	-	+14.9

1/ Experiment could not be continued because of deposition of solid amine salt.

18/ Without doubt, the extractability of amines from acid solutions depends on several factors, but only molecular weight was considered in these experiments.

19/ Hatcher, W. H., and Skirrow, F. W., The Compounds of Phenol and the Cresols with Pyridine: Jour. Am. Chem. Soc., vol. 39, 1917, pp. 1939-1977; Kester, E. B., Extraction of Phenols from Caustic Solutions: Ind. Eng. Chem., vol. 24, 1932, pp. 1121-1125.

The first section of appendix B contains data on the distribution of aniline between cresol-toluene solutions and aqueous acid. Here it is shown that when more than 15 percent of cresol is present in the toluene, relatively large amounts of aniline are removed from the aqueous layer. These data are significant in connection with the conclusions reached above concerning the difficulty of complete extraction of tar bases in the presence of tar acids.

The second section of appendix B contains a discussion of compound formation between tar acids and bases and the influence of this on the difficulty of extracting tar bases in the presence of tar acids.

Other Variables and Possible Errors in Extraction with Acid

A few experiments, using 5-, 10-, 20-, and 30-percent hydrochloric acid containing 10 percent of aniline for the extraction of a 15-percent cresol solution in toluene, showed that the acid concentration was of minor importance.

To examine the possibility that the sulphuric acid reagent would cause olefins to polymerize or to react with phenols, several reactive olefins (d-limonene, styrene, and pinene) were treated with 20-percent sulphuric acid saturated with sodium chloride. There was no loss in volume of the olefins or any indication of reaction, and on distillation the hydrocarbons came over at their respective boiling points.

ANALYTICAL PROCEDURES FOR TAR ACIDS AND BASES

Bureau of Mines Method

After analyzing several synthetic mixtures under different conditions with various acid and alkaline reagents, Kester and Rochenbach[20] adopted the method[21] described below for determining tar acids and bases. The tar oil (100 cc) is extracted with acid and alkali in the following order: (1) Three 50-cc portions of 20-percent sulphuric acid saturated with potassium sulphate, (2) three 50-cc portions of 10-percent sodium hydroxide, and (3) three 50-cc portions of 20-percent sulphuric acid saturated with potassium sulphate. This method, which employs reagents conforming to the requirements defined above (tables 3 and 4), has been used repeatedly with good results, and is interesting in that the unusual procedure of extracting tar bases first is employed.

Numerous analyses of coal-tar distillates by the Bureau of Mines method have confirmed[22] the claim of Hatcher and Skirrow[23], which is in agreement with the experience of coal-tar workers, that tar bases cannot be extracted conveniently in the presence of tar acids. Since mineral acids bring about only partial extraction of tar bases in the presence of tar acids, it seemed likely that washing with acid, then alkali, and finally with acid (as in the Bureau of

---

20/ Kester, E. B., and Rochenbach, L. P., Work cited (footnote 5).
21/ Fieldner, A. C., and Davis, J. D., Work cited (footnote 9).
22/ Communication from Messrs. J. D. Davis and L. P. Rochenbach.
23/ Hatcher, W. H., and Skirrow, F. W., The Compounds of Phenol and the Cresols with Pyridine: Jour. Am. Chem. Soc., vol. 39, 1917, pp. 1939-1977.

Mines method) would separate the tar bases roughly into two types - (1) amines having distribution ratios favorable to the acid layer and (2) amines with a distribution ratio more favorable to the hydrocarbon-phenols layer. The ratio between quantities extracted in the presence and absence of tar acids should depend on the composition of the tar oil and thus be a characteristic property. These ratios have been calculated for several tar oils; the oils from low-temperature tars yielded ratios varying between zero and 0.1, while the high-temperature tar oil (see table 8) gave a higher ratio (0.54). As might be expected from their composition, the synthetic solutions described in table 8 gave still higher ratios (1.2 to 1.4). Since high-temperature tar oils contain larger proportions of amines of low molecular weight than oils from low-temperature tars, it follows that, considering the amine ratios given above, more of the amines of low molecular weight are removed in the first acid washings, i. e., before the tar acids are removed. Experimental evidence that supports this view has been given in table 6. Cheng and Morgan[24] have shown that chiefly phenols of low molecular weight are removed when tar acids are incompletely extracted by alkali from tar oils.

In view of the results obtained by extracting amines-toluene solutions with sodium hydroxide containing cresol (see table 3), it might be expected that oils containing high concentrations of tar acids and bases could not be analyzed accurately by extraction methods. To find the maximum concentration amenable to accurate analysis by contraction methods, several synthetic mixtures were extracted by a modification of the Bureau of Mines method. Uneven portions (100, 35, and 15 cc) of the reagents were employed, and sodium chloride was used instead of potassium sulphate to saturate the 20-percent sulphuric acid. The synthetic solutions were prepared by mixing known volumes of an amine mixture (equal parts by volume of pyridine, quinoline, isoquinoline, and aniline), cresol, and toluene. The results of these analyses, given in table 7, show that the solutions containing large amounts of phenols and amines gave high results for phenols and low results for amines. However, fairly accurate data were obtained with the synthetic mixtures having the concentration of amines and phenols usually found in coal tars. In view of the fact that the solutions of low amines and phenols content gave satisfactory analyses, it might be expected that dilutions of coal oils before extraction would be beneficial[25].

It appears that the ratio of amines to phenols, as well as total content, is important. As an example, when the ratio (amines to phenols) was 1 : 1, good results were obtained with a total amines-phenols content of 40 percent; but with a ratio of 1 : 5, errors appeared with a total amines-phenols content as low as 25 percent.

24/ Cheng, Y. C., and Morgan, J. J., Work cited (footnote 7).

25/ Dilution of oils of high tar acid and base content to obtain improved analyses has been suggested by Hill[26] and by Fridli and Raffay[27].

26/ Hill, J. B., Work cited (footnote 9).

27/ Fridli, R., and Raffay, B., Determination of Creosote in Brown-Coal Tar Oils: Magyar Chem. Folyoirat, vol. 40, 1934, pp. 125-135: (Chem. Abs., vol. 29, 1935, p. 3810).

TABLE 7. - Analysis of synthetic solutions by modified Bureau of Mines method

Percent amines		Percent phenols		Percent neutral oils		Ratio amines to phenols
Calcd.	Found	Calcd.	Found	Calcd.	Found (difference)	
1	1.4	1	1.6	98	97	1:1
5	4.9	5	4.5	90	90.6	1:1
10	10.4	10	9.4	80	80.2	1:1
15	15.3	15	14.8	70	69.9	1:1
20	20.1	20	20.3	60	59.6	1:1
1	1.6	3	3.1	96	95.3	1:3
5	5.4	15	14.9	80	79.7	1:3
7	6.7	21	20.9	72	72.4	1:3
8	7.9	24	23.8	68	68.3	1:3
9	8.0	27	28.5	64	63.5	1:3
10	8.7	30	1/ 31.7	60	59.6	1:3
15	11.7	45	1/ 46.3	40	42.0	1:3
1	.8	5	5.3	94	93.9	1:5
3	2.8	15	15.5	82	81.7	1:5
4	3.7	20	20	76	76.3	1:5
5	4.0	25	26.1	70	69.9	1:5
1	1.1	7	7.4	92	91.5	1:7
3	3.1	21	20.9	76	76.0	1:7
4	3.7	28	28.4	68	67.9	1:7
5	4.3	35	36.3	60	59.4	1:7
3	3.0	27	27.3	70	69.7	1:9
4	2.8	36	37.8	60	59.4	1:9
3	2.5	33	34.4	64	63.1	1:11
4	2.5	44	47.5	52	50	1:11
2	1.8	26	26.8	72	71.4	1:13
3	1.9	39	40.3	58	57.8	1:13
1	..7	30	30.1	69	69.2	1:30
15	13.5	5	4.3	80	82.2	3:1

1/ The alkali used (150 cc of 10-percent sodium hydroxide) was not sufficient to remove all the cresol.

It will be noted (table 7) that in most instances low results were obtained for the neutral oil. This probably indicates that usually small, and in some cases considerable, amounts of the neutral oil also were extracted.

Comparison of Several Extraction Procedures

Using both synthetic solutions and coal-tar distillates, a comparison was made of the Bureau of Mines method and several other typical extraction procedures. In view of the generally satisfactory results obtained previously with the Bureau of Mines method (table 7), the data obtained with coal oils by this procedure (table 8) were considered fairly accurate, and hence of some value as a standard for evaluating the other methods. The "liberation method" of Morgan and Meighan[28] gave more accurate results for two synthetic solutions (table 8), and was therefore employed to ascertain the true tar-acid content of the coal oils.

The several oils and synthetic mixtures described in table 8 were analyzed as described below.

1. Bureau of Mines method, considered above in some detail.

2. Modified Bureau of Mines method (sodium chloride instead of potassium sulphate in 20-percent sulphuric acid, and uneven portions of reagents) used previously to obtain the data in table 7.

3. Essentially the Bureau of Mines method backward. The oil (100 cc) was extracted with three 50-cc portions of 10-percent alkali and then with three 50-cc portions of 20-percent sulphuric acid (saturated with sodium chloride). These alternate extractions were repeated until all tar acids and bases were removed.

4. A special method based on the use of 5-percent sodium hydroxide, which, in the presence of sodium phenolates, dissolves smaller quantities of amines and forms worse emulsions than 10-percent alkali. The following extraction procedure

(a) 100 cc of 5-percent sodium hydroxide.
(b) 100 cc of 5-percent sodium hydroxide.
(c) 50 cc of 10-percent sodium hydroxide.
(d) 50 cc of 20-percent sulphuric acid (sodium chloride).
(e) 25 cc of 20-percent sulphuric acid (sodium chloride).
(f) 25 cc of 20-percent sulphuric acid (sodium chloride).
(g) 25 cc of 10-percent sodium hydroxide.
(h) 25 cc of acid reagent.

was designed to benefit from the advantages of both concentrations of alkali; the bulk of the phenols was removed with 5-percent alkali, and the last traces of phenols with 10-percent alkali, which generally causes a good separation of layers. This method gave troublesome emulsions with some tar oils, and hence is of little value.

28/ Morgan, J. J., and Meighan, M. H., Work cited (footnote 4).

5. Liberation method. The oil was extracted with 20-percent alkali, the alkaline extract was acidified, the acidic solution was extracted with benzene, the benzene washings were diluted to 100 cc and extracted with 10-percent sodium hydroxide; the volume loss of the benzene layer indicated the tar acid content.

6. The volume loss of 100 cc of tar oil to 20-percent sodium hydroxide as carried out in method 5. It was observed in several cases that 20-percent alkali removed all the tar acids; that is, no further quantities of tar acids could be obtained by removing the tar bases and extracting again with alkali.

TABLE 8. - Comparison of extraction methods

Oil examined	1 B. of M.		2 Modified B. of M.		3 B. of M. backward		4 5 % NaOH method		5 Liberation method	6 20 % NaOH method	7 1/ Amines ratio
	% Amines	% Phenols	% Amines	% Phenols	% Amines	% Phenols	% Amines	% Phenols	% Phenols	% Phenols	
SM 1 ...	4.3	25.9	4.1	25.8	4.5	26.0	4.7	25.3	24.4	27.1	1.2
SM 2 ...	8.4	31.7	8.5	31.7	7.4	32.7	8.4	31.4	30.6	35.6	1.4
LT 1 ...	2.7	22.1	2.7	22.1	2.9	22.5	-	-	21.2	24.2	.0
LT 2 ...	-	20.4	-	20.4	-	21.1	-	20.6	19.2	22.4	-
LT 3 ...	2.1	22.2	2.4	21.9	2.9	22.7	-	-	21.5	23.5	.1
LT 4 ...	2.9	19.3	3.0	19.2	3.6	19.4	-	-	19.0	21.6	.07
HT 1 ...	3.7	16.1	3.7	16.3	3.6	16.8	-	-	-	-	.54
HC 1 ...	1.8	13.6	1.6	13.8	-	13.6	-	-	12.0	15.1	.29

SM 1 - Synthetic solution, 5 % amines, 25 % phenols.
SM 2 - Synthetic solution, 10 % amines, 30 % phenols.
LT 1 - 4 low-temperature tar distillates.
HT 1 - high-temperature tar distillate.
HC 1 - hydrogenated coal distillate.

1/ Ratio of amines obtained before and after removal of tar acids by Bureau of Mines method.

In general, it can be said that the results obtained with the various tar oils (table 8) are consistent with those obtained with synthetic mixtures. As might be expected, the greatest volume losses were caused by washing with 20-percent sodium hydroxide (method 6), which undoubtedly removed considerable quantities of amines and probably some of the neutral oil along with the phenols. The values afforded by the liberation method were lower than those obtained by simple extraction with 10- or 20-percent alkali; this result was to be anticipated and is in agreement with a generalization made previously by Weiss[29].

Methods 1 and 2, differing from each other only in minor details, gave almost identical results in all cases. In view of the good agreement with the values of the liberation method and the results obtained with synthetic solutions, it appears that methods 1 and 2 generally give satisfactory results. It seems likely that in most cases the true value of the tar-acid content lies between the results afforded by methods 1 (or 2) and 5. Methods 1 and 2 generally are preferable to 5 (liberation method) because of their simplicity and the fact that values are obtained for both the tar acid and base content. Method 4 was superior to the other procedures with synthetic mixtures but could not be used conveniently with the tar oils.

In a few cases (data not shown) 5- and 10-percent sodium hydroxide solutions saturated with sodium chloride were employed in extractions but appeared to have no advantage over alkali alone.

A number of facts have been established that present a rather clear picture of the Bureau of Mines method and explain the beneficial effect of extracting first with acid. The following interpretation of the Bureau of Mines type of tar acid and base analysis is consistent with the data presented thus far:

Extraction first with acid removes principally the amines of lower molecular weight and saturates the tar-oil layer with water[30]. The amines of higher molecular weight have distribution ratios more favorable to the hydrocarbon-phenols layer and hence are not removed by acid. Only negligible quantities of phenols are removed in the acid wash because (1) a suitable acid reagent (20-percent sulphuric acid saturated with an inorganic salt) is used, (2) relatively small quantities of amines are present, (3) only a portion of the amines present (certain amines of low molecular weight) as salts in acid solution is capable of extracting phenols in appreciable quantities from hydrocarbon solutions, and (4) only the phenols of low molecular weight are easily extracted by acid-amine solutions. Extraction with alkali in the second stage of the analysis removes the phenols and probably most of the water absorbed in the first washing. Possibly traces of certain highly alkylated phenols (such as thymol and phenols difficultly soluble in alkali) are not extracted. Only small quantities of amines are removed along with the sodium phenolates because (1) moderately dilute alkali is used, (2) only amines of high molecular weight, sparingly soluble in sodium phenolate solutions, are present, and (3) the amine content, relatively low at the beginning, is lower than its original value. In the third

---

29/ Weiss, J. M., Work cited (footnote 6).

30/ This is partly responsible for the low amine and high phenol values usually obtained.

stage of the analysis the remaining amines of high molecular weight are removed easily by mineral acid extraction, since phenols are not present. In several cases of the present work a second extraction with alkali, as a fourth stage of the analysis, caused a slight diminution in volume, indicating that such a procedure might be necessary in some cases to remove all traces of tar acids.

Where tar oils are extracted first with alkali virtually all the tar acids are removed, since tar bases have little retarding effect. The highly alkylated, weakly acidic phenols are less easily extracted, especially in the presence of tar bases. Considerable amounts of tar bases are removed along with the phenols, and usually the amount of water going into the oil layer is small. The amount of tar bases removed in the alkali wash is directly proportional to the concentration of tar acids, alkali, and tar bases, and inversely proportional to the molecular weight of the tar bases. The structure of the tar bases and acids also is a factor. After the tar acids have been removed, the tar bases, as is well known, are removed easily with mineral acid.

Analysis of Oils Containing Carboxylic Acids

To determine the amount, generally small, of carboxylic acids and other acids of similar strength in tar oils, the extract obtained by washing with alkali usually is saturated with carbon dioxide and then acidified with hydrochloric or sulphuric acid[31/, 32/, 33/]. The weakly acidic phenols are liberated by carbon dioxide and the stronger acids by mineral acid. Methods of this kind are time-consuming and necessitate acidification of considerable amounts of carbonate solutions. A study of synthetic solutions and tar oils containing various amounts of benzoic and acetic acids has shown that the strong tar acids may be determined directly by extraction with sodium or potassium carbonate solutions[34/]. The synthetic solutions were prepared by mixing the desired amounts of cresol, a carboxylic acid, toluene, and an amine mixture (equal parts by volume of pyridine, quinoline, isoquinoline, and aniline). The amount of carboxylic acid added was judged by the increase in volume of the solution.

31/ Morgan, G. T., Pratt, D. D., and Ross, J., Work cited (footnote 11).

32/ Stadnikov, G. L., Methods for the Analysis of Solid Fuels: Khim Tverdogo Topliva, vol. 5, 1934, pp. 193-200 (Chem. Abs., vol. 28, 1934, p. 7467).

33/ Fuel Research Technical Paper No. 37, Work cited (footnote 13).

34/ Holroyd, R., and Wheeler, R. V., Studies in the Composition of Coal. Oil-yielding Constituents: Jour. Chem. Soc., 1928, pp. 2669-2677; Karavaev, N. M., and Rapoport, I. B., Results of the Technical-Chemical Investigation of Barzass Sapromyxites: "Sapromyxites from Barzass", Goskhimizdat Petrograd, 1933, pp. 44-70 (Chem. Abs., vol. 28, 1934, p. 2158).

The solution and tar oils were analyzed by the following reagents and order of extraction.

(1) 100 cc of 20-percent sulphuric acid (sodium chloride).
(2) 50 cc of 20-percent sulphuric acid (sodium chloride).

(3) 50 cc of 10-percent sodium carbonate.

(4) 100 cc of 10-percent sodium hydroxide.
(5) 35 cc of 10-percent sodium hydroxide.
(6) 15 cc of 10-percent sodium hydroxide.

(7) 50 cc of 20-percent sulphuric acid (sodium chloride).
(8) 25 cc of 20-percent sulphuric acid (sodium chloride).
(9) 15 cc of 20-percent sulphuric acid (sodium chloride).

(10) 10-percent sodium hydroxide.

The volume of oil was recorded at extractions 2, 3, 6, 9, and 10. The last extraction was made to remove any trace of phenols not accounted for previously. In the case of the synthetic solutions the volume loss at this point was only 0.2 to 0.3 cc and was disregarded. In the case of the tar oils, appreciable volume losses often were observed in the last extraction with alkali.

The results obtained (table 9) indicate that carboxylic acids may be determined quite satisfactorily, the values being found to agree very well with the amounts present. Nothing new was revealed concerning the determination of amines and phenols. As observed in previous analyses, the amine values are low and the phenol values high.

TABLE 9. - Synthetic solutions containing benzoic acid

Amines, percent by volume				Phenols, percent by volume		Carboxylic acids, percent by volume	
Present	Found						
	First wash	Second wash	Total	Present	Found	Present	Found
5.1	3.0	1.7	4.7	20.0	20.4	0.0	0.0
5.2	2.9	1.8	4.7	20.5	21.2	.7	.7
4.6	2.6	1.7	4.3	20.1	20.9	1.2	1.1
4.8	2.7	1.8	4.5	19.9	20.2	2.6	2.7

Correction for Water Absorption

The suggestion has been made previously that absorption of water by the tar oil is partly responsible for the fact that the values obtained for amines and phenols usually are low and high, respectively. It has been observed in many instances that washing 100 cc of tar oil with 50 cc of 20-percent sulphuric acid (saturated with inorganic salt) causes the oil layer to undergo a volume increase of about 1 cc. During this operation some quantity of tar bases must have passed into the aqueous layer, hence yielding a low value for the amines content. It is likely that most of the water dissolved in the oil layer is removed in the alkaline extraction along with the phenols, resulting in a high value for the tar-acid content.

It has been found that errors arising from water absorption may be obviated somewhat by adding 2 to 3 cc of aqueous sodium chloride solution to the oil as the first stage of the analysis. The increase in volume caused by this washing was noted and later subtracted from the volume contraction to alkali. The tar-base value was indicated directly by the contraction in volume caused by washing the water-containing oil (oil washed with sodium chloride solution) with the sulphuric acid reagent.

To ascertain the effect of adding water before extracting with sulphuric acid and sodium hydroxide solutions, one synthetic solution (prepared as previously from cresol, toluene, pyridine, quinoline, isoquinoline, and aniline) and several tar-oil distillates were analyzed by a procedure identical with that used to obtain the data in table 9, with the exception that a preliminary wash with 2 to 3 cc of sodium chloride solution was employed. It was found necessary to allow considerable time for the layers to separate when the oil was washed with carbonate solutions. In some instances a nearly saturated solution of potassium carbonate was less troublesome than 10-percent sodium carbonate, and with either reagent a good separation of the layers must be effected to determine traces of carboxylic acids. For the purpose of comparison, the same synthetic solution and tar oils (see table 10) were analyzed by the modified Bureau of Mines method and the liberation procedure of Morgan and Meighan (methods 2 and 5 in table 8). Small amounts of benzoic acid were added to the tar oils numbered 5 and 8, but in all instances the liberation method was carried out on the original untreated distillates.

As may be seen from the data in table 10, washing the oil with salt solution before extracting the tar acids and bases caused higher and lower values, respectively, to be obtained for the amines and phenols. Such values are more accurate than those obtained by simple extraction methods; hence, the preliminary wash with salt water is a worthy addition to the contraction method.

TABLE 10. - Analysis for acidic and basic constituents with preliminary addition of salt-water solution.

Oil no.	1 Modified Bureau of Mines method		2 Salt-water solution added			3 Liberation method
	Amines,%	Phenols,%	Amines,%	Phenols,%	Carboxylic acids,%	Phenols,%
1.	2.6	37.9	3.6	37.0	0.0	-
2.	-	-	2.7	13.4	1.0	13.0
3.	1.7	27.3	2.2	26.8	.0	26.5
4.	1.9	28.9	2.0	28.6	.0	26.7
5.	-	-	2.4	31.8	1.6	30.0
6.	5.5	19.7	5.2	19.4	.0	19.4
7.	6.2	26.1	6.3	24.9	.6	25.4
8.	2.8	15.1	3.4	12.6	1.2	14.1

1. Synthetic solution: 4% amines, 36% cresol, and 60% toluene; 0.9 cc water absorbed.
2. Hydrogenated coal distillate; 0.2 cc water absorbed.
3. Coal tar (500° carbonizing temp.) distillate; 0.6 cc water absorbed.
4. Coal tar (600° carbonizing temp.) distillate; .4 cc water absorbed.
5. Coal tar (700° carbonizing temp.) distillate; .6 cc water absorbed. 1.2% of benzoic acid added.
6. Coal tar (800° carbonizing temp.) distillate; 0.2 cc water absorbed.
7. Coal tar (800° carbonizing temp.) distillate: .3 cc water absorbed.
8. Coal tar (900° carbonizing temp.) distillate; .4 cc water absorbed. 0.8% benzoic acid added.

1.-4. 10-percent sodium carbonate used to remove carboxylic acids.
5 -8. Almost saturated potassium carbonate solution used to remove carboxylic acids.

The data in table 11 show that oils containing appreciable amounts of water-soluble amines and carboxylic acids give erroneous results when analyzed by the procedure described previously and used to obtain the data in table 10. This is due to the fact that the water-soluble carboxylic acid (acetic acid) is extracted along with the amines in the sulphuric acid wash. More accurate results (see solutions 3 and 4 in table 11) were obtained by extracting the acetic acid just after the sodium chloride-water treatment and before removal of the amines.

TABLE 11. - Analyses of synthetic solutions[1/] containing acetic acid

Synthetic solution no.	Water absorbed, cc	Amines, volume percent		Phenols, volume percent		Carboxylic acids, volume percent	
		Present	Found	Present	Found	Present	Found
1	0.4	5.2	6.3	20.4	20.6	1.6	0.5
2	.5	5.3	7.5	20.4	20.5	2.8	.7
3	.5	5.2	5.2	20.1	19.9	2.0	2.6
4	.4	5.5	5.0	20.0	20.2	1.6	2.1

1/ Synthetic solutions prepared from toluene, cresol, acetic acid, and an amine mixture (equal parts by volume of pyridine, quinoline, isoquinoline, and aniline).

1. Carboxylic acid extracted with saturated potassium carbonate after sulphuric acid extraction.
2. Carboxylic acid extracted with 10-percent sodium carbonate after sulphuric acid extraction.
3. Carboxylic acid extracted with saturated potassium carbonate after sodium chloride-water treatment.
4. Carboxylic acid extracted with 10-percent sodium carbonate after sodium chloride-water treatment.

Procedure Recommended for Determination of Tar Acids and Bases

The procedure described below is recommended for determining tar bases, carboxylic acids, and phenols in tar oils (using about 100 cc of oil in a graduated separatory funnel[35/]). The second step, extraction with potassium carbonate solution, may be omitted if it is not desired to determine carboxylic acids separately.

1. The oil is shaken with 2 to 3 cc of an approximately saturated sodium chloride solution; the increased volume of the oil layer is noted and used subsequently in calculating volume contractions.

35/ Weiss, J. M., Work cited (footnote 6).

2. Extracted with 5 to 10 cc of approximately saturated potassium carbonate solution; diminution in volume of oil layer is indication of carboxylic acid content.

3. Extracted in succession with 100- and 50-cc portions of the sulphuric acid reagent (20-percent sulphuric acid approximately saturated with sodium chloride).

4. Extracted in succession with 100-, 35-, and 15-cc portions of 10-percent sodium hydroxide.

5. Extracted in succession with 50-, 25-, and 15-cc portions of the sulphuric acid reagent.

6. Extracted with 15 to 20 cc of 10-percent sodium hydroxide.

The amounts of tar bases and phenols present are indicated by the total volume contractions to sulphuric acid and sodium hydroxide, respectively. The tar-acid value is corrected by subtracting from it the water absorption value (obtained in the first step).

SUMMARY AND CONCLUSIONS

Neither dilute nor concentrated alkali is capable of extracting appreciable quantities of amines from hydrocarbon solutions. However, concentrated alkali containing considerable amounts of phenol sodium salts may extract large amounts of amines and smaller amounts of neutral oil from hydrocarbon solutions. Therefore, concentrated alkali is not a suitable reagent for the analytical extraction of tar acids. Dilute alkali also is objectionable as an analytical reagent because of its tendency to form emulsions with tar distillates. Aqueous solutions containing about 10 percent of sodium hydroxide extract only negligible amounts of nonacidic material under favorable conditions, have little tendency to form emulsions, and hence are suitable reagents for extracting tar acids.

The solubility of tar bases in aqueous sodium phenolate solutions depends upon the structure and molecular weight of the tar base. The ring-nitrogen bases (pyridine, quinoline) are more soluble than the bases of equal molecular weight with the nitrogen attached to the ring (aniline and its derivatives). The former type, ring-nitrogen bases, is more abundant in coal-tar distillates. Amines of low molecular weight are more soluble in sodium phenolate solutions than amines of high molecular weight and similar structure. For these reasons, the amines removed from tar oils by extraction with alkali are chiefly those of low molecular weight, probably pyridine and its derivatives.

Since the tar bases of low molecular weight are most soluble in alkaline phenolate solutions, it is desirable in analytical procedures to remove these bases before extracting with alkali. The amines removed from tar oils by mineral acid extraction are preponderantly those of low molecular weight. Accordingly, more accurate tar acid and base analyses may be obtained by washing with the acid reagent before carrying out the alkaline extraction. This conclusion is supported also by experiments in which tar distillates were analyzed by various extraction procedures.

The amount of tar bases removed from tar oils by alkaline extraction is directly proportional to the concentration of both tar acids and bases in the oil. Hence, tar oils containing high concentrations of tar acids and bases are more likely to give erroneous results on analysis.

Although some dilute mineral acid solutions extract appreciable quantities of phenols of low molecular weight from hydrocarbon solutions, 20-percent sulphuric acid saturated with certain inorganic salts (preferably sodium chloride) extracts only negligible amounts. This reagent (20-percent sulphuric acid saturated with sodium chloride) failed to attach several sensitive unsaturated hydrocarbons and has been found satisfactory in the analytical extraction of tar bases.

Under favorable conditions, dilute mineral acids containing amine salts may extract considerable amounts of phenols from hydrocarbon solutions. The amount of phenols extracted in this manner is directly proportional to the concentration of amine salts and inversely proportional to the amine salt molecular weight. In this connection the concentration of the mineral acid has little effect.

It has been shown by analyzing synthetic solutions of amines and phenols in toluene that the Bureau of Mines method of determining tar acids and bases gives good results for the lower concentrations usually found in coal-tar distillates. Since more satisfactory analyses were obtained with the synthetic solutions containing relatively small amounts of amines and phenols, it is to be expected that dilution of tar oils of high tar acid and base content with a suitable solvent prior to analysis would be beneficial. Such a procedure has been recommended previously.

It has been observed that the ratio of tar bases extracted by mineral acids before and after removal of the tar acids depends upon the composition of the tar oil. This ratio is higher for distillates from high-temperature tars than for those from low-temperature tars.

Several typical extraction procedures have been compared, using synthetic solutions, coal tar, and hydrogenated coal distillates. This study showed that the Bureau of Mines type of extraction (tar bases extracted first) gives results that are quite accurate and in fair agreement with values obtained by the "liberation method". The liberation method gave lower tar-acid values than the simple contraction methods.

It has been shown that the determination of tar acids may be modified to give separate values for the phenols and carboxylic acids. The carboxylic acids are removed by extraction with sodium or potassium carbonate and the more weakly acidic phenols are later extracted with sodium hydroxide.

The high and low results, respectively, obtained for tar acids and bases are partly due to absorption of water by the tar oil in the first extraction. A method for correcting this error is described.

A modification of the present Bureau of Mines method for analyzing tar oils, which includes a correction for water absorption and separate determinations of phenols and carboxylic acids, is recommended. This modified method is believed to be simpler and more accurate than the methods previously suggested for determining tar bases, phenols, and carboxylic acids.

The nature of the hydrocarbon solvent was found to be important, more accurate extractions being obtained with an aromatic solvent, toluene. Accordingly, aromatic solvents should be used when tar oils are diluted prior to analysis.

Data are given in appendix A which show that the structure and molecular weight of the sodium phenolate in aqueous solution have an important effect on the amount of amines passing from the oil into the aqueous layer.

A study of the difficulty of extracting tar bases with mineral acids in the presence of tar acids is presented in appendix B. It is shown that the concentration of phenols in the tar oil plays a significant role and that compound formation between the tar acids and bases is of minor importance.

APPENDIX A. IMPORTANT VARIABLES AND POSSIBLE ERRORS IN EXTRACTION WITH ALKALI

Concentration of Amine in Toluene Solution

In the usual manner, toluene solutions (100 cc) containing various concentrations of aniline were washed several times with 50-cc portions of an alkaline-cresol solution (30 cc of cresol added to 100 cc of 20-percent sodium hydroxide). Pure toluene was not appreciably soluble in this alkaline-cresol solution, but, as shown in table 12, aniline-toluene solutions were quite soluble. The diminution in volume of the aniline-toluene solution was directly proportional to the aniline concentration. This is in agreement with the work of Hatcher and Skirrow[36], who found that the amount of pyridine extracted by alkali from pyridine-benzene solutions containing a constant amount of phenol was directly proportional to the concentration of pyridine.

TABLE 12. - Extraction of aniline-toluene solutions with alkaline-cresol solution

Concentration of aniline, percent by volume	Volume change of toluene layer, cc			
	1st 50 cc	2d 50 cc	3d 50 cc	1st 150 cc
0	+0.1	-0.3	0.0	-0.2
15	-1.9	-2.1	-1.8	-5.8
25	-3.0	-3.6	-3.0	-9.6
40	-4.8	-5.3	-4.9	-15.0
55	-7.0	-7.6	-6.9	-21.5

36/ Hatcher, W. H., and Skirrow, F. W., Work cited (footnote 19).

Hydrocarbon Solvent

Only two solvents were examined, and in this comparison it was found that the solvent plays an important part. In tar oils, of course, the neutral oil is a complex mixture of various types of hydrocarbon, but one oil may be more "aromatic" in nature than another. Pure toluene and Eastman's "heptane", b.p. 91°-96°, were the hydrocarbons compared in the present work. Solutions (25 percent amine by volume) of xylidine in toluene and in heptane were washed in the usual manner with an alkaline-cresol solution (30 cc of cresol added to 100 cc of 20-percent sodium hydroxide). It was found (see table 3) that the loss in volume of the xylidine-heptane solution was considerably greater than the volume loss of the xylidine-toluene solution. This result was not unexpected in view of the fact that aromatic amines are more soluble in aromatic than in paraffinic hydrocarbons. It seems safe to conclude from these experiments that in analytical extractions the errors will be greater if the hydrocarbon solvent is paraffinic in nature and that aromatic solvents should be used when tar oils are diluted before analysis.

TABLE 13. - Extraction of xylidine-toluene and xylidine-heptane solutions with alkaline-cresol solution

25-percent xylidine solution in -	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
Toluene	-1.0	-1.4	-1.2	-1.2	-3.6
	-1.0	-1.3	-1.1	-1.3	-3.4
Heptane	-4.0	-3.5	-3.2	-	-10.7
	-4.0	-3.5	-3.2	-	-10.7

Molecular Weight of Sodium Phenolate

The standard amines-toluene solution (15.35 percent by volume of amines) described on page 4 was washed successively with 50-cc portions of alkaline-phenol solutions, which had been prepared by adding 15.5 g of the desired phenol to 100 cc of 20-percent sodium hydroxide. The phenols not highly alkylated (phenol, cresol, naphthol) removed amines from the toluene layers in quantities directly proportional to the molecular weight of the phenol. Anaylsis of the remaining portions of the amines-toluene layers by extraction with 10-percent sodium hydroxide showed that no appreciable amount of the phenol had passed from the aqueous into the toluene layer in the phenol, cresol, and naphthol experiments. It is obvious (see table 14) that the amines-toluene solution extracted the phenols from the alkaline solutions of thymol and butyl-phenol. It had been shown[37] previously that thymol and alkylated phenols are easily extracted from caustic solutions with organic solvents and that such

37/ Kester, E. B., Work cited (footnote 19).

extractions are facilitated by the addition of amines[38] to the organic solvent. While no definite conclusion can be drawn from the present work, it appears that molecular structure or some factor other than molecular weight plays an important role. This is readily apparent upon comparison of the results obtained with thymol and sec.-butylphenol with those for the other phenols of table 14.

The amines-toluene solution used to obtain the data in table 14 was found to be more effective than toluene in extracting thymol from sodium hydroxide solution (15.5 g thymol added to 100 cc of 20-percent sodium hydroxide). Washing 100 cc of the amines-toluene solution with 50 cc of the alkaline-thymol solution caused the organic layer to undergo a volume increase of 6.8 cc (see table 14). Where 100 cc of toluene was washed in a similar manner the volume increase was 1.9 cc.

TABLE 14. - Extraction of amines-toluene with alkaline-phenols solution (15.5 g of the phenol in 100 cc of 20 percent NaOH)

Phenol and mol. wgt.	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
Phenol 96	0.0 .0	-0.3 - .3	-0.2 - .2	-	-0.5 - .5
Cresol 130	-1.1 -1.2	-1.5 -1.5	-1.3 -	-	-3.9 -
Alpha naphthol 144	-3.6	-3.5	-2.5	-	-9.6
Thymol 150	+6.8	[1] +16.0	-	-	-
Sec.-butylphenol 150	[1] +14.5 [1] +13.9				

[1] 3 layers formed; the volume reported is that of the 2 upper layers.

In connection with the solubility of amines in sodium phenolates of different molecular weight, it is interesting to note that Cheng and Morgan[39] have shown that aqueous solutions of sodium phenolates of moderately high or intermediate molecular weights have the greatest solvent action for tar acids.

38/ Drees, K., and Kowalski, G., Die Extraktion von Phenolen aus Phenolatlaugen: Brennstoff-Chem., vol. 14, 1933, pp. 382-383: See also E. B. Koster (to Koppers Gas and Coke Co.), U. S. Patent 2,043,102, June 2, 1936 (Chem. Abs., vol. 30, 1936, p. 4873).

39/ Cheng, Y. C., and Morgan, J. J., Work cited (footnote 7).

APPENDIX B

Distribution of Aniline Between Cresol-Toluene and Aqueous Acid Solutions

Several cresol-toluene solutions containing 0, 7.5, 15, 22.5, and 30 percent of cresol were studied to determine the influence of phenol concentration. These solutions were washed with a standard aniline-hydrochloric acid solution, which had been prepared by adding 30 cc of aniline to 100 cc of 20-percent hydrochloric acid. The toluene solutions containing only moderate amounts of cresol suffered volume losses when washed by 50 cc portions of the aniline-hydrochloric acid solution, while the toluene solutions containing more than about 15 percent of cresol gained in volume. The volume increases of the more concentrated cresol solutions were found to be proportional to the amount of cresol in the toluene. These results are given in table 15.

After extraction with the aniline-hydrochloric acid solution, the toluene layer of the last experiment of table 15 was analyzed to determine the cresol and aniline content. Successive extractions with 10-percent sodium hydroxide and with 20-percent hydrochloric acid indicated that, although large quantities of aniline had entered the toluene layer, only negligible amounts of cresol had been extracted by the aniline-hydrochloric acid solution.

These results indicate that the difficulty of extracting tar bases with mineral acid in the presence of the tar acids would be greater for tar oils containing large amounts of tar acids.

TABLE 15. - Extraction of various cresol-toluene solutions with aniline-hydrochloric acid solution[1/] (30 cc of aniline and 100 cc of 20-percent HCl)

Concn. of cresol, percent	Volume changes of toluene layer, cc				
	1st 50 cc	2d 50 cc	3d 50 cc	4th 50 cc	1st 150 cc
7.5	-0.7	-1.1	-1.5	-	-3.3
	-	-	-	-	-
15	-1.0	-2.2	-1.9	-1.7	-5.1
	- .7	-2.3	-2.2	-1.8	-5.2
22.5	+4.2	-3.8	-7.2	-	-6.8
	-	-	-	-	-
30	+10.0	+5.2	.0	-	+15.2
	+10.1	+5.3	+ .1	-	+15.5

1/ The volume change caused by washing pure toluene with this solution was negligible.

Compound Formation Between Tar Acids and Bases

Skirrow and his students[40/, 41/], have confirmed the experience of others in showing that mineral acids extract tar bases very inefficiently in the presence of tar acids. Skirrow carried out extraction experiments with phenol-pyridine-benzene solutions to find an explanation for this phenomenon and found that, while the presence of pyridine does slightly diminish the extraction of phenol, the presence of phenol diminishes the extraction of pyridine to a much greater extent. It was postulated[42/] that the difficulty of extracting amines in the presence of phenols is due to the effect of complexes similar to II on equilibria between various compounds of amines and phenols.

$$C_6H_5OH.C_5H_5N \rightleftharpoons C_6H_5OH + C_5H_5N.$$

"If we add excess pyridine, the effect in depressing the dissociation will be dependent directly on the concentration of the pyridine. If we add excess of phenol the effect in the first instance will similarly be dependent on the concentration of phenol, but excess of phenol differs from excess of pyridine in that it promotes a second reaction,

$$C_6H_5OH.C_5H_5N + C_6H_5OH \rightleftharpoons (C_6H_5OH)_2.C_5H_5N ,$$

I II

and thus not only depresses the original dissociation in a normal way but also removes the compound $C_6H_5OH.C_5H_5N$ from the left-hand side of that equilibrium, thus still further depressing the dissociation and removing the free pyridine from the system[43/]".

This hardly can be the sole explanation, since the common amines and phenols in most cases (pyridine with o-cresol[44/], guaiacol[45/]; aniline with m-cresol[46/], p-cresol[47/], beta-naphthol[48/], guaiacol[49/]; o-toluidine with

---

40/ Hatcher, W. H., and Skirrow, F. W., Work cited (footnote 19).
41/ Skirrow, F. W., and Binmore, T. V., The Compounds of a Phenol and the Cresols with Pyridine. II: Jour. Am. Chem. Soc., vol.40,1918,pp.1431-1442.
42/ Skirrow, F. W., and Binmore, T. V., Work cited (footnote 41).
43/ Skirrow, F. W., and Binmore, T. V., Work cited (footnote 41).
44/ Bramley, A., The Study of Binary Mixtures. Part III. Freezing Point Curves: Jour. Chem. Soc., vol. 109, 1916, pp. 469-496.
45/ Pushin, N.A., and Vaic, B., Equilibrium in Binary Systems which Contain Guaiacol as One Component: Monatsh., vol. 47, 1927, pp. 529-535.
46/ Kremann, R., and Borjanovics, V., Energy Changes in Binary Systems. V. Constitution of the Ternary System in - Cresol - Aniline - Benzene According to Viscosity Measurements: Monatsh., vol.37, 1916, pp. 59-84.
47/ Philip, J. C., Freezing-Point Curves for Some Binary Mixtures of Organic Substances, Chiefly Phenols and Amines: Jour. Chem. Soc., vol. 83, 1903, pp. 814-834.
48/ Kremann, R., Lupfer, E., and Zawadsky, O., The Influence of Substitution in the Components upon the Equilibria in Binary Solutions. XXVII. The Binary Systems of M- and P- Amino- Phenols with Phenols and Nitro Compounds: Monatsh., vol. 41, 1921, pp. 499-542.
49/ Pushin, N.A., and Vaic, B., Work cited (footnote 45).

guaiacol[50]; p-toluidine with phenol[51], o-cresol[52], m-cresol[53], p-cresol[54], alpha-naphthol[55], beta-naphthol[56], guaiacol[57]; dimethylaniline with p-cresol[58]; alpha-naphthylamine with phenol[59], o-cresol[60], m-cresol[61], p-cresol[62]; beta-naphthylamine[63] with o-cresol, m-cresol, p-cresol) form only compounds (similar to I) consisting of equimolar quantities of the amine and phenol. Among the more common amines and phenols compounds similar to II are found only in a few instances (pyridine[64] with phenol and p-cresol; quinoline[65] with o-cresol, p-cresol; piperdine[66] and p-cresol). It has been claimed[67] that dimethylaniline and p-cresol do not combine to form a compound, and that aniline[68] and alpha-naphthol form two compounds. Quinoline[69] and phenol

$$C_6H_5NH_2 \cdot C_{10}H_7OH \qquad (C_6H_5NH_2)_2 \cdot C_{10}H_7OH$$

also form two compounds, as determined by freezing-point data.

$$C_9H_7N \cdot (C_6H_5OH)_2 \qquad (C_9H_7N)_3 \cdot (C_6H_5OH)_2$$

It appeared likely that the behavior of phenols and amines during extraction is largely independent of the type of compound formed, and to obtain information on this subject several extractions[70] using appropriate amines and phenols were carried out. The solutions (100 cc) contained certain percentages by volume of the amine and phenol in benzene and were extracted with slightly more than one equivalent of 10-percent sodium hydroxide and 20-percent hydrochloric acid (based on the original content of amine and phenol). The results are shown in table 16.

50/ Pushin, N.A., and Vaic, B., Work cited (footnote 45).
51/ Philip, J. C., Work cited (footnote 47).
52/ Pushin, N. A., and Sladovic, L., Equilibrium in the Binary Systems Cresols-Amines: Jour. Chem. Soc., 1928, pp. 2474-2483.
53/ Pushin, N. A., and Sladovic, L., Work cited (footnote 51).
54/ Pushin, N. A., and Sladovic, L., Work cited (footnote 51).
55/ Philip, J. C., Work cited (footnote 47).
56/ Kremann, R., Lupfer, E., and Zawadsky, O., Work cited (footnote 48).
57/ Pushin, N.A., and Vaic, B., Work cited (footnote 45).
58/ Pushin, N.A. and Vaic, B., Work cited (footnote 45).
59/ Philip, J. C., Work cited (footnote 47).
60/ Pushin, N. A., and Basara, D., Equilibrium in Binary Systems Containing as One Component a Cresol: Monatsh., vol. 48, 1927, pp. 51-60.
61/ Pushin, N. A., and Basara, D., Work cited (footnote 60).
62/ Pushin, N. A., and Basara, D., Work cited (footnote 60).
63/ Pushin, N. A., and Basara, D., Work cited (footnote 60).
64/ Bramley, A., Work cited (footnote 44).
65/ Pushin, N. A., and Sladovic, L., Work cited (footnote 52).
66/ Pushin, N. A., and Sladovic, L., Work cited (footnote 52).
67/ Pushin, N. A., and Sladovic, L., Work cited (footnote 52).
68/ Kremann, R., Lupfer, E., and Zawadsky, O., Work cited (footnote 48).
69/ Bramley, A., Work cited (footnote 44).
70/ It was pointed out by Skirrow and Binmore[71] that such experiments would yield useful information, but as yet no description of work of this kind has appeared.
71/ Skirrow, F.W., and Binmore, T. V., Work cited (footnote 41).

TABLE 16. - Extraction of synthetic solutions with NaOH and HCl

Extracted with -	Volume loss of toluene layer, cc										
	1	2	3	4	5	6	7	8	9	10	11
NaOH	-	-	4.7	5.1	-	5.6	-	0.6	-	3.2	--
HCl	1.2	0.4	19.9	20.0	1/	19.3	4.9	21.3	-	20.9	--
NaOH	21.4	23.2	.2	-	23.9	.3	-	2.2	-	1.3	--
HCl	2.0	2.1	.1	-	2.9	.0	-	.6	-	.2	--
NaOH	.9	.7	-	-	.2	-	-	1.3	2.3	-	4.2
NaOH	-	-	-	-	-	-	-	.7	1.4	-	1.1
NaOH	-	-	-	-	-	-	-	.2	.8	-	.2
NaOH	-	-	-	-	-	-	-	-	.6	-	-

Solution	Amine, percent by volume	Phenol, percent by volume	Molecular compounds, amines to phenols	Extracted with	
				20% HCl, cc portions	10% NaOH, cc portions
1	Pyridine, 5	o-cresol, 20	1:1	15	110
2	do.	p-cresol, 20	1:1 1:2	15	110
3	Aniline, 20	Beta-naphthol, 5	1:1	50	30
4	do.	Alpha-naphthol, 5	1:1 2:1	50	30
5	Aniline, 5	p-cresol, 21	1:1	15	110
6	Aniline, 20	p-cresol, 5.4	1:1	50	25
7	Aniline, 5	0.0	---	15	---
8	Aniline, 20	Thymol, 5	---	50	30
9	0.0	do.	---	--	30
10	Aniline, 20	1,3,2-xylenol, 5	---	50	30
11	0.0	do.	---	--	30

1/ Gained 1.6 cc in volume.

It will be noted that, although the only compound formed is the 1:1 type, the extraction of pyridine is retarded by o-cresol (solution 1, table 16). The extraction of pyridine is retarded more by p-cresol, and it is possible that this difference between the effects of o- and p-cresol may be explained by Skirrow and Binmore's hypothesis[72]. The extraction of the alpha-and beta-naphthol solutions was of interest because alpha-naphthol forms two compounds ($C_{10}H_7OH.C_6H_5NH_2$ and $C_{10}H_7OH.(C_6H_5NH_2)_2$) with aniline, while beta-naphthol forms only the 1:1 compound. On the basis of the 2:1 compound and Skirrow and Binmore's theory, difficulty might be expected in extracting alpha-naphthol in the presence of aniline. Actually, both naphthols were easily extracted with alkali in the presence of a large excess of aniline. Aniline and p-cresol form only the 1:1 compound, but it was found that alkali removed the cresol easily and that acid caused a volume increase (solutions 5 and 6). Under similar conditions, aniline was extracted without difficulty from benzene (solution 7). It was to be predicted that thymol and 2,6-dimethylphenol, which are easily extracted from their caustic solutions by organic solvents, would be extracted with more difficulty than the usual phenols from either hydrocarbon or amines-hydrocarbon solutions. It may be realized from the data in table 16 that this is true and that it is more difficult to extract such phenols in the presence of amines. This is in agreement with Hatcher and Skirrow's observation[73] that pyridine diminishes slightly the extraction of phenol, and with Drees and Kowalski's claim[74] that amine-hydrocarbon solutions are more effective than hydrocarbon solvents in extracting phenols from alkaline solutions.

On the basis of the experiments described in table 16, it can be said that amines diminish slightly the extraction of phenols and that the extent of this phenomenon depends upon the nature of the phenol and amine and on other factors. This effect is not easily observed when, as in tar oils, the tar base content is low, and may not occur, due to the solvent action of concentrated alkaline-phenols solutions for both amines and phenols, when the phenols are extracted with concentrated alkali. Phenols diminish the extraction of amines to a much greater extent, and this effect is observed easily in tar oils, the tar-acid content of which usually is considerably higher than the tar-base content. It seems probable that compounds of the type

$$\text{phenol.(amine)}_2$$

play only a minor role in extraction phenomena, and that the effects observed are determined chiefly by the distribution ratios of amines and phenols between the various solutions involved.

72/ Skirrow, F. W., and Binmore, T. V., Work cited (footnote 41).
73/ Hatcher, W. H., and Skirrow, F. W., Work cited (footnote 23).
74/ Drees, K., and Kowalski, G., Work cited (footnote 38).

After this report has served your purpose and if you have no further need for it, please return it to the Bureau of Mines. The use of this mailing label to do so will be official business and no postage stamps will be required.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

622.09
Un32rmo
Cop2

RI 3311

SEPTEMBER 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

COOPERATIVE FUEL RESEARCH MOTOR-GASOLINE SURVEY, WINTER 1935-36

The work upon which this report was based was performed under a cooperative agreement between the Bureau of Mines, Department of the Interior, and the Cooperative Fuel Research Committee

COMPILED BY

E. C. LANE AND A. J. KRAEMER

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3311,
September 1936.

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

COOPERATIVE FUEL RESEARCH MOTOR-GASOLINE SURVEY, WINTER 1935-36[1]

Compiled by E. C. Lane[2] and A. J. Kraemer[3]

INTRODUCTION

This is the first publication in a projected series of reports on properties of commercial motor fuels pursuant to a cooperative agreement between the Cooperative Fuel Research (C.F.R.) Committee and the Bureau of Mines of the Department of the Interior. The report gives results of analyses in oil-company laboratories of samples of gasoline obtained from service-station pumps in 18 marketing areas by representatives of oil companies. These companies obtained the samples and analyzed them for their own information and supplied data from their laboratory reports to the Bureau of Mines for compilation and publication.

History of Bureau of Mines Gasoline Surveys

The following quotation from Report of Investigations 3162[4] is included in this place for the information of readers who are not familiar with the Bureau of Mines gasoline surveys that have preceded the present report:

> The Bureau of Mines made its first survey of motor gasolines sold throughout the United States in 1915,[5] a second in 1917,[6] and a third in April 1919.[6] These three surveys showed that during each 2-year interval the motor gasoline had become notably less

---

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3311."

2/ Assistant petroleum chemist, U.S. Bureau of Mines.

3/ Senior refinery engineer, U.S. Bureau of Mines.

4/ Kraemer, A. J., and Lane, E. C., Motor-Gasoline Survey, August 1931, Part I-Specification Data: Rept. of Investigations 3162, Bureau of Mines, 1932, 23 pp.
Kraemer, A. J., Lane, E. C., and Garton, E. L., Motor-Gasoline Survey, August 1931, Part II - Additional Data, with a section on Detonation Characteristics of Motor Fuels, by N. R. White and H. K. Cummings of the National Bureau of Standards: Rept. of Investigations 3175, Bureau of Mines, 1932, 31 pp.

5/ Rittman, W. F., Jacobs, W. A., and Dean, E. W., Physical and Chemical Properties of Gasoline Sold Throughout the United States During the Calendar Year 1915: Tech. Paper 163, Bureau of Mines, 1916, 45 pp. (out of print; consult libraries).

6/ Hill, H. H., and Dean, E. W., Quality of Gasoline Marketed in the United States: Bull. 191, Bureau of Mines, 1921, 275 pp. (out of print; consult libraries).

volatile. In order to ascertain whether the change in volatility was continuing, a series of semiannual surveys was instituted. The first of these semiannual series was made in January 1920.1/ These semiannual surveys have been continued without interruption since that time.

During the period covered by these surveys a great deal of study and experimental investigation has been carried on by various research groups and individuals throughout the industry to determine the essential properties of motor fuels in terms of performance in engines operating on the explosive cycle.

Knowledge of these requisites of performance is now much more general than was any technical knowledge of gasoline in 1915. This increased technical knowledge has been applied practically in manufacturing processes and has aided in bringing about a degree of uniformity of motor fuel properties. The American motorist is now afforded a plentiful supply of a relatively low-priced fuel, available even in the most remote sections of the country, and motor fuel which is probably better suited to the requirements of the automobile engine than any fuels that were generally available when the earlier gasoline surveys of the Bureau of Mines were made.

The twenty-first semiannual survey (January 1930) gave evidence that a more complicated motor-gasoline situation had been developing within the United States. A larger variety of motor fuels and a wider range of retail prices were encountered by Bureau of Mines engineers in collecting the samples. An increased emphasis on premium quality was noted also. To make the twenty-second semiannual survey (August 1930) more comprehensive than those that had preceded it, the Bureau of Mines decided to test the supposition that local conditions in those sections of the United States not reached directly by the past several surveys were different from the conditions indicated by the samples that had been collected. Therefore, samples were taken in 19 areas instead of 12 as had been done in several previous surveys, and the number of samples was doubled; approximately 300 samples were taken as compared with approximately 150 in the past. For comparison, the samples were divided into "competitive-price gasoline" and "premium-price motor fuels." Some investigators had felt that the practice followed in previous semiannual surveys of grouping all types of motor fuel together without differentiating between "competitive-price gasoline"; benzol blends, Ethyl gasoline, "high-test" gasoline, and other "premium" fuels had detracted from the usefulness of the survey reports. They felt that the data did not permit drawing conclusions in regard to any one type of fuel and tended to obscure differences between fuels of the various types.

1/ Smith, N. A. C., The Motor-Gasoline Surveys of 1920 and 1921: Tech. Paper 328, Bureau of Mines, 1923, 41 pp.

Inception of the New Plan

After the results of the survey of August 1931 were published the Bureau was compelled to suspend its gasoline surveys due to lack of funds and has not been able to resume the work on the former basis.

Lack of current data of the kind contained in the survey of August 1931[8/] prompted representatives of the automotive and petroleum industries through the C. F. R. Committee to propose to the Bureau of Mines a method of compiling gasoline surveys according to a plan somewhat different from the method of previous surveys that would be less costly to the Bureau of Mines and could be carried on within its restricted appropriations. The basis of the new proposal was the fact that various oil companies throughout the United States make periodical surveys of motor fuels sold in the areas in which their own products are sold. These company surveys contain the same data as had been reported in Bureau of Mines gasoline surveys. It was thought that these company surveys might give a more intensive coverage of the gasolines sold in a given city because each company would be in a position to analyze a greater number of brands of gasoline in the cities in which it was interested as a marketer. Therefore the suggestion was made that the companies supply data from their reports on samples from various cities to the Bureau of Mines so that the Bureau could compile surveys without incurring the expense and loss of time involved in taking samples throughout the United States and analyzing them in its laboratories. The C. F. R. Committee acted as the coordinating agency under this plan.

This committee was organized in 1921 as a joint project of the National Automobile Chamber of Commerce (now the Automobile Manufacturers Association), the American Petroleum Institute, the Society of Automotive Engineers, and the National Bureau of Standards to study problems involved in the mutual adaptation of the fuel and the engine to each other to the end of national economy and internal-combustion-engine efficiency.[9/]

The committee proposed that the United States Bureau of Mines compile, tabulate, analyze, and comment upon data furnished by individual companies and publish a report of the study for free distribution. After studying all of the considerations involved, the Bureau of Mines assented to this proposal, and a cooperative agreement was signed by representatives of both parties. The present report is the first publication under this agreement.

The following companies contributed directly to this gasoline survey: Associated Oil Co., Continental Oil Co., Empire Oil & Refining Co., Ethyl Gasoline Corporation, Gulf Refining Co., Phillips Petroleum Co., Shell Oil Co., Shell Petroleum Corporation, Sinclair Refining Co., Standard Oil Co. (Indiana), Standard Oil Development Co., Standard Oil Co. of California, Sun Oil Co., The Atlantic Refining Co., The Pure Oil Co., Tide Water Oil Co., and the Union Oil Co. of California.

8/ See footnote 4.

9/ Dickinson, H. C., The Cooperative Fuel Research and Its Results: Jour. Soc. Auto. Eng., August 1929.

The committee and the Bureau thought that a report should be published as soon as possible, on the basis of available data, although recognizing that these data are not all that could be desired. Although such a course may result in minor discrepancies, the committee and the Bureau believed that this possibility would not be serious enough to outweigh the desirability of an early report.

To expedite compilation by the Bureau of Mines of the data submitted by the cooperating companies, report forms were supplied to each of the companies by the secretary of the C.F.R. Committee so that data would be submitted in a uniform manner. The reports from the companies were tabulated at the Petroleum Experiment Station of the U.S. Bureau of Mines at Bartlesville, Okla.

The Bureau of Mines, during the initial development of the plan under which this survey has been made, intended to obtain reports from at least two of the cooperating companies on gasolines sold in a given area, so as to have a check on the characteristics reported on the various brands of gasoline. However, for the present survey reports were received for some areas from only one company. Some of the companies in their own surveys had not made a practice of determining all of the data requested on the cooperative-report form. Also, since some of these companies had already collected samples for their winter survey when the plan for a cooperative gasoline survey was broached, reports from the various companies relate to samples taken from November 1935 to February 1936. Data were not available on gasolines sold in Atlanta, Ga., and Dallas and El Paso, Tex., and consequently these cities, which had been included in previous Bureau of Mines surveys, are not included in the present survey. It is hoped that data for these cities will be available for the next survey.

The reader will note that the data on gasoline sold in the Pacific coast cities, Seattle, Wash., and Los Angeles and San Francisco, Calif., are reported in a different form from the data from other cities included in this survey. The data for the Pacific coast cities were supplied by a Pacific coast technical group made up of technologists employed by the major oil companies of California. These samples were not identified by brand name in submitting the data to the Bureau of Mines, and any given letter in the code used does not necessarily identify the product of the same company for each of the three grades. The form of the data submitted from the Pacific coast area does not permit comparison by the Bureau of Mines of analyses by two or more laboratories on the same brand of gasoline. However, the technical group has expressed the opinion that the reports for the Pacific coast marketing areas are typical for the various gasolines sold in those areas and that they give a correct picture of gasoline quality in the various areas.

The company supplying figures for the San Francisco Bay region does not make a practice of recording temperatures at which 30 percent and 70 percent are recovered in the distillation analysis, and data on these points consequently are omitted in reports on samples from the San Francisco Bay region.

It is hoped that in future cooperative motor-gasoline surveys the various discrepancies will be eliminated, and more uniform practice can be arranged respecting the time samples will be taken, number of companies reporting in each area, extent of the data to be reported, and other details of collecting and analyzing samples, and reporting results. As has been stated, for this first survey after a lapse of 4 1/2 years, the primary consideration was thought to be prompt publication of as much information as possible.

EXPLANATION OF TABLES

Table 1 gives data on gravity, sulphur content, Reid vapor pressure, copper-strip corrosion test, ASTM octane number, and distillation characteristics for commercial gasolines sold in 18 marketing areas in the United States during the winter of 1935-36. Item numbers are assigned to each brand in each area as a convenience in referring to the data. An item number in one part of the table has no relation to the same item number in another part of the table. Figures relating to each item are the averages of the data reported by co-operating companies on the brand of gasoline represented by the item number. Gravities were reported by the companies according to the A.P.I. scale:

$$\text{Degrees A.P.I.} = \frac{141.5}{\text{sp.gr. } 60^\circ/60^\circ \text{ F.}} - 131.5.$$

The specific gravity corresponding to the numerical average of the API gravities has been inserted by the Bureau of Mines. Averages for each of the grades of gasoline are included at the foot of the sections of table 1 giving data for each city.

The averages of table 1 have been assembled in tables 2, 3, and 4, which give summaries of the data of this survey. Tables 5 and 6 give for comparison data from the Bureau of Mines survey of March 1931,[10] which was the last winter gasoline survey made by the Bureau of Mines on the former basis. Table 7 gives comparisons of averages of the sulphur contents of samples from different cities in the present survey and corresponding data for the same cities in the surveys of March and August 1931.[11] Table 8 gives the same type of information regarding Reid vapor pressures in the three surveys, and table 9 gives the data on octane numbers. Table 10 summarizes the data of the present survey by stating the minimum and the maximum figure for each characteristic of each of the three groups of gasolines and giving the range of figures which covers 90 percent of the items in each group of gasolines.

10/ Lane, E. C., Garton, E. L., and Kraemer, A. J., Twenty-third Semi-Annual Motor-Gasoline Survey. Part 1: Rept. of Investigations 3129, Bureau of Mines, 1931, 20 pp.; Lane, E. C., Twenty-Third Semiannual Motor-Gasoline Survey. Part II. Specification Data: Rept. of Investigations 3142, Bureau of Mines, 1931, 13 pp.; Kraemer, A. J., Garton, E. L., and Lane, E. C., Twenty-Third Semiannual Motor-Gasoline Survey, Part III. With a section on Detonation Characteristics of Motor Fuels, by N. R. White and H. K. Cummings: Rept. of Investigations 3152, Bureau of Mines, 1931, 35 pp.

11/ Work cited. See footnote 4.

Methods Used

It is assumed that the several laboratories followed the customary procedures described in the following methods of the American Society for Testing Materials, in obtaining the data included in this report:

Sulphur: Tentative Method of Test for Sulfur in Petroleum by Lamp Method. A.S.T.M. Designation D 90 - 34 T.

Reid Vapor Pressure: Tentative Method of Test for Vapor Pressure of Motor and Aviation Gasoline (Reid Method). A.S.T.M. Designation D 417 - 35 T.

Copper Strip Test: Standard Method of Test for Detection of Free Sulfur and Corrosive Sulfur Compounds in Gasoline: A.S.T.M. Designation D 130 - 30.

A.S.T.M. Octane Number: Tentative Method of Test for Knock Characteristics of Motor Fuels: A.S.T.M. Designation D 357 - 34 T.

Distillation Range: Standard Method of Test for Distillation of Gasoline, Naphtha, Kerosine, and Similar Petroleum Products: A.S.T.M. Designation D 86 - 35.

The American Society for Testing Materials has published a discussion of the significance of these and other tests usually applied to petroleum products.12/

Distillation data are reported in terms of temperature in degrees Fahrenheit at which 10 percent, 20 percent, etc., by volume of the 100.ml charge to the distilling flask are recovered in the receiving graduate. In planning the survey members of the subcommittee stated that almost all oil-company laboratories report results of analyses for their own use on that basis and that therefore results on the basis of percent evaporated, for example, are not readily understandable. On the usual assumption that all of the distillation loss occurs before 10 percent has been recovered in the receiving graduate, the reported data can be converted to the percent evaporated basis by adding the distillation loss to the percent recovered. For example, with reference to item 1 of the regular price gasolines from Boston, it is stated in table 1 that 10 percent was recovered at a thermometer reading of 124°F. and that the distillation loss (uncondensed portion of the charge) was 1.8 percent. This is equivalent to a statement that 11.8 percent (10 percent + 1.8 percent) had been evaporated when the thermometer reading reached 124°F. Similarly, 20 percent was recovered in the graduated receiver

12/ American Society for Testing Materials, The Significance of Tests of Petroleum Products: Philadelphia, 1934, 76 pp.

when the thermometer reading was 155°F., and these data, together with the distillation loss, indicate that 21.8 percent had been evaporated when the thermometer reading was 155°F.

DISCUSSION OF RESULTS

Octane Number

Octane ratings first were included in Bureau of Mines gasoline surveys when octane numbers determined by the National Bureau of Standards were reported on composite samples from each city in the survey of August 1930.[13] Octane numbers were determined by the National Bureau of Standards also for each sample in the survey of March 1931[14] and the survey of August 1931.[15]

These surveys definitely indicated a steady increase in octane rating of commercial gasoline. In the survey of August 1930 the average octane number of composite samples of competitive-price gasolines (which included a few third-grade samples) was reported as 59.2, and for premium-price fuels the average octane number of composite samples was 73.7. The report on the survey of March 1931 contains a table[16] giving comparisons of average octane numbers of samples from the different cities of the surveys of August 1930 and March 1931, which shows a definite increase in octane ratings.

The present survey shows further increases in octane numbers, particularly of regular-price and third-grade gasolines. Probably the most striking revelation is the remarkably small range of the octane numbers of the great majority of the regular-price gasolines sold throughout the United States. Table 2 shows that the average octane numbers for different cities during the winter of 1935-36 range from 68.6 for Seattle to 70.2 for San Francisco. Octane ratings of regular-price gasolines for which this characteristic have been reported range from 61 to 72, and 90 percent have octane numbers of 68 to 71 inclusive.

Table 9 shows that average octane numbers of regular-price gasolines sold in different cities are markedly higher for the present survey than average octane numbers of competitive price gasolines of the two previous surveys of Bureau of Mines, on samples collected in August and March 1931, respectively. Although the competitive price groups in the 1931 surveys include third-grade gasolines, which lowered the average octane number of the group, inspection of the octane numbers of individual samples in those surveys indicate plainly that the octane numbers of regular-price gasolines in 1931 were lower than during the winter of 1935-36. This is indicated by the circumstance that only 42 of the 207 samples (15.5 percent) of competitive price gasolines in the March 1931 survey had an octane number of 69 or higher.

13/ Kraemer, A. J. and Lane, E. C., Twenty-second Semiannual Motor-Gasoline Survey. Additional Data: Rept. of Investigations 3092, Bureau of Mines, 1931, pp. 11-14.

14/ See footnote 10.

15/ See footnote 4.

16/ Rept. of Investigations 3152, table 10. See footnote 10.

Octane numbers of premium-price gasolines in the present survey range from 69.1 to 81, and 90 percent have octane numbers of 73.7 to 80.5 inclusive, a narrower range than was shown by the survey of March 1931. This circumstance apparently is due mainly to three developments during the interval of time between the two surveys: (1) A change in the method for determining octane number was made in 1933. (2) A fairly large number of samples classed as premium-price gasolines in the survey of March 1931 were sold at only a slight advance over the regular-price gasolines, and these gasolines generally had relatively low octane numbers. (3) Octane numbers of 80 or higher are reported only for Seattle, Los Angeles, and San Francisco in the present survey, whereas in the survey of March 1931, 31 of the 105 samples of premium-price motor fuel had octane numbers of 80 or higher, and these samples were almost equally divided between the three Pacific coast cities and cities in the eastern portion of the country.

Octane numbers of third-grade gasolines cover a wider range than either of the two other grades in this survey ranging from 33.5 to 69.1. Ninety percent have octane numbers of 50 to 66 inclusive. Octane numbers typical of regular-price gasolines have been recorded for some brands of third-grade gasolines; and, on the other hand, octane numbers of 52 or below have been reported for 24 brands of third-grade gasoline. Octane ratings of third-grade gasolines apparently are fairly uniform in some areas, for example, New York, Omaha and Lincoln, Seattle, and Los Angeles. In other areas the range is rather wide, as in Chicago, for example. In the Eastern Seaboard cities and on the Pacific coast octane ratings of third-grade gasolines usually are high, averaging 63. In Cleveland, St. Louis, Tulsa, Denver, and other cities of the Middle West, octane ratings of third-grade gasolines are noticeably lower, averaging approximately 52.

In general the spread between the average octane numbers of third-grade and regular-price gasolines is greater than the spread between the average octane numbers of the regular-price and the premium-price gasolines. Individual instances contrary to this generalization can be found in table 1. The uniformity of the octane numbers of most of the regular-price gasolines throughout the United States has been noted, as well as the somewhat wider range of octane numbers of the majority of the premium-price gasolines. Octane ratings of third-grade gasolines cover a considerably wider range, and a smaller proportion of the entire number of third-grade gasolines have octane numbers within a narrow range of values.

Volatility

Volatility of gasoline is indicated by the Reid vapor pressure and the distillation data, and only to a minor degree by the gravity.

Reid vapor pressure.- Regular-price gasolines for which this characteristic has been reported in the present survey have Reid vapor pressures from 5.8 to 13.15 pounds per square inch, and 90 percent are between 7.1 and 12.4 pounds, inclusive.

For premium-price gasoline, reported Reid vapor pressures range from 3.1 to 12.55 pounds, and 90 percent are between 5.0 and 11.5 pounds, inclusive.

Reid vapor pressures of third-grade gasolines for which this characteristic has been reported range from 4.2 to 13.4 pounds per square inch, and 90 percent are between 5.5 and 10.4 pounds, inclusive.

Table 8 shows that the average Reid vapor pressures of regular-price gasoline sold during the winter of 1935-36 in different areas included in this survey were higher than for competitive-price gasolines in the same areas in March 1931 and that the average Reid vapor pressures of premium-price gasolines were nearly the same in the cities in the winter of 1935-36 as in March 1931. Average Reid vapor pressures of third-grade gasolines were lower than for the two other grades.

Table 1 shows a rather wide variation in Reid vapor pressures of individual brands of regular-price and premium-price gasoline in the same city. For example in Boston the Reid vapor pressure of item 2 of the regular-price gasolines is 6.8 pounds and for item 1 the pressure is 12.4 pounds per square inch. Similarly, in Cleveland, two of the premium-price gasolines have vapor pressures of 7.0 and 7.6, and two others have vapor pressures of 11.5 and 11.6 pounds per square inch. Vapor pressures below 8 pounds per square inch are more usual among the third-grade gasolines.

Distillation data.- Table 10 gives minimum and maximum temperatures for various po ints in the distillation ranges of the three groups of gasolines, together with the range of temperatures that covers 90 percent of the items in each group. These data illustrate the distillation characteristics of gasolines sold during the winter of 1935-36. Changes in distillation characteristics during the interval between March 1931 and the winter of 1935-36 are indicated by the data on distillation range in tables 5 and 6, in comparison with similar data in tables 2, 3, and 4. The greater volatility of gasolines sold during the winter of 1935-36 probably is shown best by comparison of the figures in the columns for 20 percent recovered and 50 percent recovered in these tables. These figures are definitely lower for the present survey than corresponding figures for the survey of March 1931, indicating greater volatility.

Gravity.- Comparison of the data on gravities of samples in this report with the averages of gravities of samples from different cities in the survey of March 1931 shows that on the whole, gasolines were lighter in gravity during the winter of 1935-36 than in March 1931. Until comparatively recently, gravity was considered to be probably the most important characteristic of motor gasoline. In recent years other standards of comparison have been adopted for judging the comparative quality of motor gasolines, and gravity is now considered to be a minor characteristic, having only an indirect relationship to the essential qualities of gasoline. Knowledge of the gravity of a gasoline in relation to other characteristics such as vapor pressure, octane number, and distillation range is an aid in forming judgments regarding the source or type of gasoline. Gravity also has some bearing on the thermal value of a gasoline.

Sulphur Content

The sulphur content of gasolines is measured by two different types of test: (1) The copper-strip corrosion test, and (2) the lamp method for total sulphur content.

Copper-strip test.- Only two of the gasolines in this survey were reported as failing to pass the test, and only four of more than 1,000 samples of gasoline that have been subjected to this test in the laboratories of the Bureau of Mines since January 1930 have failed to pass.

Total sulphur content.- Table 1 shows that the sulphur content of the regular-price gasolines for which sulphur content has been reported ranges from 0.014 to 0.30 percent; 90 percent of the regular-price gasolines have sulphur content between 0.02 and 0.15 percent, inclusive.

The sulphur content of premium-price gasolines for which sulphur content has been reported ranges from 0.006 to 0.179 percent; 90 percent of the premium-price gasolines have sulphur content between 0.02 and 0.08 inclusive.

Sulphur contents that have been reported for third-grade gasolines range from 0.016 to 0.23 percent; 90 percent have sulphur content between 0.02 and 0.20 percent, inclusive.

Data included in this survey indicate that the highest figures for sulphur contents are reported for the regular-price and third-grade gasolines sold in the Pacific coast marketing areas and the lowest sulphur contents in the Mid-Continent areas. Premium-price gasolines included in this survey contain little sulphur, and the average sulphur content of third-grade gasolines is slightly lower than for regular price gasolines.

SUMMARY

This survey of motor gasoline sold to the general public through service stations in 18 marketing areas throughout the United States during the winter of 1935-36 is the first that has been published pursuant to a cooperative agreement between the United States Bureau of Mines and the Cooperative Fuel Research Committee.

Characteristics of regular-price, premium-price, and third-grade motor gasolines are reported, based upon analyses made in oil company laboratories and submitted to the Bureau of Mines for compilation and comment.

This survey discloses that, insofar as the characteristics included in this survey serve to indicate quality in gasoline, the progressive improvement in gasoline sold to the general public that had been shown in previous motor gasoline surveys made by the Bureau of Mines has continued.

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

BOSTON, MASS.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper strip test	ASTM octane number
	Specific	°API				
1	0.723	64.2	0.09	12.4	Neg.	70.6
2	.728	62.9	.03	6.8	do.	69.4
3	.735	61.0	.05	10.3	do.	69.5
4	.732	61.8	.08	9.1	do.	69.5
5	.734	61.3	.08	9.5	do.	69.0
6	.732	61.8	.04	9.5	do.	68.0
7	.731	62.1	.11	11.1	do.	68.7
8	.735	61.0	.04	10.0	do.	69.5
9	.738	60.2	.07	10.4	do.	68.5
10	.735	61.0	.08	10.0	do.	70.4
11	.741	59.5	.07	9.3	do.	69.6
Average	.733	61.5	.07	9.9	do.	69.3

Premium-price gasoline

1	0.722	64.5	0.03	10.0	Neg.	77.7
2	.779	50.1	.07	7.7	do.	75.0
3	.740	59.7	.08	10.0	do.	76.7
4	.714	66.7	.03	9.5	do.	75.5
5	.716	66.1	.02	9.4	do.	78.2
Average	.739	60.0	.05	9.3	do.	76.6

Third-grade gasoline

1	0.729	62.6	-	9.4	Neg.	61
2	.732	61.7	-	10.5	-	64
3	.735	61.1	-	10.6	Neg.	65
4	.739	60.0	-	9.8	do.	65
5	.744	58.7	-	8.9	do.	65
6	.750	57.2	-	6.9	Pos.	59.
7	.734	61.2	0.03	8.8	Neg.	63.4
Average	.738	60.2	-	9.3	do.	62.9

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

BOSTON, MASS.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
83	124	155	184	229	270	328	385	2.2	1.8	1
88	134	163	191	237	278	344	407	1.1	1.7	2
88	130	162	191	236	274	337	385	2.0	1.6	3
92	140	168	195	239	276	342	389	1.8	1.4	4
91	137	167	214	241	284	345	391	1.3	1.5	5
87	142	173	199	243	284	344	398	1.8	1.5	6
89	129	156	181	226	269	313	388	1.6	1.1	7
99	149	178	203	247	289	348	400	3.1	1.9	8
90	135	163	192	240	283	350	402	1.7	1.6	9
90	127	154	182	240	291	355	398	1.2	1.3	10
94	138	167	199	248	288	350	396	1.8	2.0	11
90	135	164	194	239	281	341	394	1.8	1.6	Average

Premium-price gasoline (continued)

91	134	157	179	218	257	320	395	1.0	1.5	1
108	150	168	179	198	235	318	383	1.1	1.2	2
91	135	162	192	245	288	359	383	1.0	1.8	3
97	138	157	173	200	227	271	335	1.1	1.7	4
95	138	158	175	202	228	274	339	1.0	1.8	5
96	139	160	180	213	247	308	367	1.0	1.6	Average

Third-grade gasoline (continued)

90	133	160	185	230	267	330	385	3.0	1.0	1
86	140	174	202	244	276	328	377	1.0	3.0	2
95	139	164	187	223	261	320	387	2.0	1.0	3
93	137	166	194	243	282	342	392	3.0	1.0	4
86	131	165	196	252	298	356	414	2.0	1.0	5
90	151	185	214	266	314	374	430	2.0	1.0	6
87	145	174	201	246	291	351	416	1.2	1.8	7
90	139	170	197	243	284	343	400	2.0	1.4	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

NEW YORK, N.Y.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.738	60.2	0.086	11.11	Neg.	69.5
2	.736	60.8	.068	10.6	do.	70.0
3	.732	61.8	.075	10.2	do.	70.1
4	.731	62.1	.115	10.05	do.	70.6
5	.739	60.0	.075	9.7	do.	70.3
6	.724	63.9	.048	11.0	do.	70.0
7	.738	60.2	.066	8.5	do.	69.8
8	.731	62.1	.07	10.8	do.	68.3
9	.734	61.3	.032	8.9	do.	69.0
10	.733	61.5	.030	10.05	do.	69.5
Average	.734	61.3	.067	10.01	do.	69.7

Premium-price gasoline

1	0.736	60.8	0.031	8.6	Neg.	76.7
2	.740	59.7	.072	9.7	do.	77.6
3	.718	65.6	.041	11.1	do.	77.8
4	.734	61.3	.08	9.7	do.	76.4
5	.733	61.5	.064	9.3	do.	76.9
6	.717	65.9	.030	9.2	do.	79.6
7	.729	62.6	.100	10.7	do.	76.0
Average	.730	62.3	.06	9.8	do.	77.6

Third-grade gasoline

1	0.727	63.0	-	-	-	64.9
2	.742	59.1	0.083	10.1	-	67.5
3	.743	58.8	.084	9.5	-	66.9
4	.752	56.6	-	6.8	-	63
Average	.741	59.4	-	8.8	-	65.6

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

NEW YORK, N.Y.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
87	130	161	191	242	289	356	402	1.3	2.7	1
86	125	153	183	241	293	353	394	.9	1.8	2
85	129	159	188	236	279	346	389	1.3	1.9	3
88	130	158	187	237	284	346	408	1.6	1.6	4
86	133	165	196	247	291	351	394	1.5	1.7	5
84	124	154	182	235	279	333	381	1.4	1.6	6
95	144	174	202	246	288	345	409	1.8	1.4	7
86	130	157	183	247	269	334	386	1.5	1.8	8
94	141	168	195	239	284	348	397	1.1	1.7	9
94	138	168	193	246	296	353	395	1.1	1.7	10
89	132	162	190	242	285	347	396	1.4	1.8	Average

Premium-price gasoline (continued)

93	144	176	203	250	293	351	395	1.2	1.1	1
92	131	157	188	244	292	351	394	1.3	1.7	2
89	129	152	172	210	249	319	391	1.6	1.8	3
90	134	165	194	240	286	350	393	1.2	1.9	4
94	137	163	190	232	270	330	376	1.5	1.4	5
97	138	164	174	203	232	278	360	1.0	1.6	6
96	137	164	191	231	272	341	391	.8	2.5	7
93	136	163	187	230	271	331	386	1.2	1.7	Average

Third-grade gasoline (continued)

84	136	165	192	235	275	338	387	0.8	3.2	1
88	133	169	199	253	295	352	408	.6	1.9	2
86	133	171	203	254	297	349	421	1.0	1.5	3
97	152	185	215	268	316	365	407	.9	1.7	4
89	139	172	202	252	296	351	406	.8	2.1	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

WASHINGTON, D.C. (including samples taken in Baltimore)

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.729	62.5	0.083	9.9	Neg.	70.4
2	.732	61.8	.03	9.3	do.	69.6
3	.735	61.0	.053	8.7	do.	69.3
4	.738	60.2	.081	10.7	do.	69.4
5	.737	60.5	.057	9.9	do.	69.3
6	.734	61.3	.095	9.8	do.	68.3
7	.740	59.7	.077	11.1	do.	68.7
8	.734	61.3	.058	8.9	do.	68.9
Average	.735	61.0	.067	9.8	do.	69.2

Premium-price gasoline

1	0.740	59.6	-	8.3	-	74
2	.778	50.4	0.108	9.0	Neg.	76.4
3	.720	65.0	.037	10.3	do.	79
4	.736	60.7	.075	7.2	do.	-
5	.717	65.9	.045	9.4	do.	-
6	.740	59.7	.067	9.9	do.	-
7	.777	50.7	.179	8.5	do.	-
8	.738	60.2	.042	8.2	do.	-
Average	.747	57.9	.069	8.9	do.	76.4

Third-grade gasoline

1	0.738	60.2	0.056	-	Neg.	69.
2	.728	62.9	.10	10.2	do.	66.1
3	.756	55.6	.143	8.6	do.	69.1
4	.745	58.4	.08	8.0	do.	58.4
Average	.742	59.2	.095	8.9	do.	65.6

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

WASHINGTON, D.C. (including samples taken in Baltimore)

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
92	129	154	193	242	287	348	396	1.1	1.8	1
90	136	165	193	240	278	343	397	1.1	1.4	2
92	134	165	193	247	294	353	395	1.0	2.0	3
84	130	161	192	242	287	357	402	1.0	2.2	4
86	125	153	183	240	291	351	393	1.0	1.8	5
87	131	159	183	228	270	337	391	1.1	1.6	6
88	135	164	193	245	287	347	390	1.0	1.7	7
90	136	166	194	247	293	355	395	1.0	2.0	8
89	132	161	191	241	286	349	395	1.0	1.8	Average

Premium-price gasoline (continued)

91	140	172	200	247	294	351	392	0.7	1.4	1
99	143	165	182	211	268	359	396	.8	1.8	2
90	133	155	175	212	250	322	394	1.2	2.1	3
101	156	178	198	234	271	335	394	1.1	.9	4
99	139	158	174	207	237	297	364	1.3	1.2	5
91	134	162	194	248	298	354	393	1.2	1.8	6
103	149	165	178	198	234	324	385	1.1	1.4	7
99	152	178	204	248	293	355	401	1.1	1.4	8
97	143	167	188	226	268	337	390	1.1	1.5	Average

Third-grade gasoline (continued)

86	122	152	182	242	299	362	396	1.	2.	1
78	122	151	182	234	277	329	388	1.1	1.9	2
92	136	160	183	241	283	352	405	1.1	1.4	3
83	136	168	206	255	293	346	402	1.0	2.0	4
85	129	158	188	243	288	347	398	1.1	1.8	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

PITTSBURGH, PA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.729	62.6	0.063	9.8	Neg.	70.7
2	.740	59.7	.082	10.3	do.	69.8
3	.730	62.3	.060	9.4	do.	68.7
4	.739	60.0	.068	9.8	do.	70.9
5	.728	62.9	.045	10.1	do.	70.0
6	.737	60.5	.056	11.7	do.	69.5
7	.728	62.9	.062	10.6	do.	69.0
8	.724	63.9	.041	-	do.	70.0
9	.736	60.7	.03	-	do.	66
10	.729	62.6	.084	9.0	do.	70.7
11	.723	64.2	.053	9.5	do.	-
12	.727	63.0	.086	9.7	do.	-
13	.728	62.9	.057	9.8	do.	70.7
Average	.731	62.1	.061	10.0	do.	70.0

Premium-price gasoline

1	0.711	67.5	0.032	9.4	Neg.	77.9
2	.735	61.0	.049	9.7	do.	75
3	.727	63.0	.03	9.8	do.	77.
4	.723	64.1	.026	-	-	76.3
5	.719	65.2	.031	9.8	Neg.	-
6	.722	64.6	.036	9.4	do.	-
7	.729	62.6	.055	10.4	do.	-
8	.739	59.9	.044	9.0	do.	-
9	.740	59.8	.051	9.2	do.	-
10	.773	51.6	.070	8.8	do.	-
Average	.732	61.8	.042	9.5	do.	76.6

TABLE 1.- Motor-gasoline survey, Winter, 1935-36 - Continued
(Data for samples from different cities)

PITTSBURGH, PA.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
90	134	161	188	236	278	340	402	1.2	2.0	1
88	133	164	196	248	293	360	405	1.1	2.0	2
86	135	166	193	241	286	352	405	.9	2.2	3
87	134	165	197	251	296	355	393	1.0	2.0	4
88	133	161	189	239	284	349	403	1.1	2.2	5
84	129	159	193	246	288	351	391	1.0	2.8	6
87	130	157	187	238	285	352	403	1.0	2.0	7
88	149	183	210	251	289	353	415	1.0	2.3	8
92	134	163	192	240	288	346	399	1	1	9
92	131	155	182	232	275	337	387	1.0	2.1	10
92	155	185	209	248	277	330	358	1.0	2.0	11
93	136	163	187	232	275	333	379	1.0	1.5	12
85	135	152	190	237	279	346	396	1.0	2.0	13
89	136	164	193	242	284	347	395	1.0	2.0	Average

Premium-price gasoline (continued)

96	138	156	171	200	225	266	331	1.1	1.9	1
94	138	163	187	226	263	312	357	1.1	1.7	2
90	118	151	175	222	268	330	400	.9	2.1	3
90	133	158	180	223	268	329	385	.3	1.7	4
100	144	164	184	213	246	298	345	1.0	1.4	5
93	141	164	185	222	256	310	367	1.2	1.8	6
88	138	166	194	241	279	349	397	1.0	2.0	7
95	140	169	196	247	295	369	417	1.1	1.4	8
91	138	166	198	258	305	351	387	1.0	1.5	9
100	147	163	175	190	217	298	365	1.2	1.3	10
94	137	162	184	224	262	321	375	.9	1.6	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

CLEVELAND, OHIO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.731	62.1	0.048	9.8	Neg.	70.1
2	.732	61.8	.043	9.7	do.	69.4
3	.731	62.1	.100	12.7	do.	69.4
4	.726	63.4	.045	11.7	do.	70.8
5	.733	61.5	.04	9.5	do.	69.5
6	.739	60.0	-	10.3	do.	70.0
7	.726	63.4	.052	10.0	do.	70.1
8	.734	61.3	.093	9.9	do.	69.5
9	.738	60.2	.069	9.9	do.	70.0
10	.722	64.5	.050	11.6	do.	70.6
Average	.731	62.1	.06	10.5	do.	69.9

Premium-price gasoline

1	0.713	67.0	0.024	11.6	Neg.	76.7
2	.723	64.2	.030	7.6	do.	76
3	.741	59.5	-	-	-	78
4	.728	62.9	.03	10.2	Neg.	76.3
5	.717	65.9	.016	11.5	do.	76
6	.722	64.6	-	7.0	do.	75.0
7	.720	65.0	.02	10.2	do.	75.2
Average	.723	64.2	.024	9.7	do.	76.2

Third-grade gasoline

1	0.737	60.5	0.018	6.5	Neg.	48.8
2	.736	60.8	.065	6.3	do.	50.1
3	.735	61.1	-	6.0	do.	54
4	.723	64.3	-	8.0	do.	53.5
5	.730	62.2	.042	6.5	do.	47
6	.740	59.7	.025	6.5	do.	50.3
Average	.733	61.5	.038	6.6	do.	50.6

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

CLEVELAND, OHIO.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
88	132	160	187	236	285	365	417	1.1	2.4	1
87	135	166	194	245	290	352	401	.9	2.0	2
82	122	150	177	226	279	361	393	1.0	4.0	3
88	126	154	181	227	273	366	439	1.1	2.9	4
96	137	172	200	250	297	363	403	1.0	2.5	5
84	134	166	194	245	290	346	394	.6	3.0	6
89	131	163	190	236	276	345	397	.8	2.7	7
90	134	164	193	236	282	343	395	1.0	2.5	8
96	140	168	200	248	296	358	398	1.0	2.0	9
83	120	144	171	229	282	360	410	1.0	2.0	10
88	131	161	189	238	285	356	405	1.0	2.6	Average

Premium-price gasoline (continued)

85	126	150	176	219	256	327	400	1.2	2.9	1
98	147	170	189	222	250	300	363	.8	1.8	2
90	132	164	193	242	288	351	395	.7	1.9	3
92	138	168	196	232	273	339	376	1.0	3.6	4
89	128	156	184	230	273	336	394	1.1	2.9	5
99	143	169	190	223	254	304	362	1.0	1.0	6
82	125	158	185	227	268	340	404	1.1	2.2	7
91	134	162	187	228	266	328	385	1.0	2.3	Average

Third-grade gasoline (continued)

106	166	196	219	256	298	353	396	1.0	1.5	1
102	161	191	215	253	292	356	418	.8	1.7	2
111	153	177	199	237	288	350	397	1.0	1.0	3
99	152	186	214	249	288	337	374	1.0	2.5	4
102	164	194	220	257	296	346	396	1.2	1.8	5
101	157	186	213	261	312	375	419	.9	1.7	6
104	159	188	213	252	279	353	400	1.0	1.7	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

CHICAGO, ILL.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.708	68.4	0.039	12.1	Neg.	70.0
2	.716	66.1	.04	11.9	do.	70.1
3	.720	65.0	.025	11.2	do.	69.5
4	.718	65.6	.054	12.6	do.	70.2
5	.720	65.0	.052	11.4	do.	70.7
6	.726	63.4	.031	11.3	do.	69.7
7	.721	64.8	.044	12.3	do.	69.0
8	.720	65.0	.034	10.6	-	70.3
9	.721	64.8	.037	11.8	do.	69.9
10	.724	63.9	-	11.1	do.	71.0
11	.723	64.2	.033	11.4	do.	69.6
12	.729	62.6	.027	10.9	do.	68.8
13	.726	63.4	.037	11.1	Pos.	69.4
14	.728	62.9	.043	10.4	Neg.	68.9
15	.732	61.8	.130	12.3	do.	69.2
16	.733	61.5	.026	10.8	do.	69.3
Average	.723	64.2	.043	11.5	do.	69.7

Premium-price gasoline

1	0.684	75.4	0.058	11.2	Neg.	75.7
2	.705	69.2	.040	10.5	do.	77.6
3	.711	67.5	.034	11.6	do.	77.5
4	.722	64.5	.024	11.8	do.	77.5
5	.735	61.0	.031	10.1	do.	73.4
6	.725	63.7	.025	10.6	do.	76.0
7	.727	63.1	.032	10.0	do.	74.0
8	.725	61.0	.029	10.7	do.	74.0
9	.734	61.3	.083	10.5	do.	76.2
10	.738	60.2	.040	9.0	do.	75.9
11	.719	65.3	-	11.65	-	77
12	.729	62.5	.048	10.4	-	77.5
13	.733	61.5	-	-	-	75.0
14	.734	61.3	.031	8.8	Neg.	72.9
15	.751	57.0	.044	3.1	do.	73.0
Average	.725	63.7	.04	10.0	do.	75.5

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

CHICAGO, ILL.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
85	110	126	144	193	256	354	412	1.0	1.9	1
85	113	136	161	216	276	359	420	.8	3.0	2
81	114	140	168	233	287	356	395	1.0	2.7	3
86	114	137	162	225	286	356	402	1.0	4.0	4
85	122	150	168	213	267	348	402	1.0	2.6	5
84	114	138	164	235	288	361	408	1.0	2.7	6
91	122	147	177	221	278	352	397	1.0	3.2	7
84	119	145	175	233	281	349	398	1.0	2.3	8
87	125	155	184	234	287	356	396	1.0	2.2	9
87	120	147	174	237	294	362	392	1.0	3.2	10
80	123	154	189	229	285	359	403	1.0	2.6	11
86	122	147	177	245	294	360	396	1.0	3.0	12
93	128	155	182	234	290	359	414	1.1	2.5	13
84	120	148	177	235	285	355	405	1.0	1.3	14
90	129	161	193	227	280	360	388	.9	4.2	15
86	119	146	173	229	283	356	389	1.0	3.1	16
							401	1.0	2.8	Average

Premium-price gasoline (continued)

93	110	117	124	144	183	267	392	.9	1.4	1
89	112	126	141	175	225	313	385	1.0	1.5	2
87	121	143	161	198	232	296	361	1.0	2.6	3
84	117	145	178	240	287	342	379	1.0	2.9	4
88	130	162	194	248	299	360	395	1.0	2.7	5
85	129	148	174	228	281	347	393	1.0	2.3	6
89	125	153	186	241	289	345	404	1.0	1.8	7
86	121	147	173	228	282	357	405	1.0	2.0	8
90	133	162	190	236	283	352	390	.9	2.6	9
90	128	159	189	248	301	363	406	1.0	2.3	10
82	115	138	162	219	276	353	417	1.0	2.7	11
91	130	158	188	235	280	347	414	1.0	2.0	12
91.5	123	151	180	243	306	378	414	1.0	2.0	13
102	141	164	188	234	287	352	404	1.0	.5	14
128	178	201	221	261	301	360	437	1.0	1.0	15
92	128	152	177	225	274	342	400	1.0	2.0	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

CHICAGO, ILL.

Third-grade gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper, strip test	ASTM octane number
	Specific	°API				
1	0.726	63.4	0.032	8.6	Neg.	51.3
2	.735	61.0	.042	7.1	do.	48.9
3	.733	61.5	.036	9.7	Pos.	58.4
4	.734	61.3	.033	9.1	Neg.	56.8
5	.733	61.5	.038	8.4	do.	52.4
6	.740	59.7	.028	8.1	do.	54.5
7	.741	59.5	.02	7.8	do.	61.8
8	.727	63.1	.035	9.6	do.	68.6
9	.728	62.8	.03	10.2	do.	65.0
10	.732	61.7	.043	10.4	do.	60.2
11	.743	59.0	.02	4.2	do.	45.0
Average	.734	61.3	.032	8.5	do.	56.6

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

CHICAGO, ILL.

Third-grade gasoline (continued)

Distillation range, °F.									
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
90	136	168	192	239	283	342	398	1.0	2.2	1
101	160	190	215	252	292	345	394	1.1	1.4	2
93	140	169	198	246	296	365	400	.9	2.8	3
92	145	181	202	249	293	358	419	1.0	2.2	4
95	139	168	198	251	303	373	422	1.1	1.2	5
92	145	184	215	264	313	370	410	1.0	1.7	6
100	149	183	211	255	298	358	410	1.0	1.8	7
100	141	156	185	241	294	356	437	1.0	1.5	8
102	137	166	193	242	294	356	437	1.0	1.5	9
102	146	176	202	249	289	349	400	1.0	1.0	10
123	176	202	223	263	303	359	409	1.0	.5	11
99	147	177	204	250	296	357	412	1.0	1.6	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

DETROIT, MICH.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.717	65.9	0.050	10.5	Neg.	69.8
2	.712	67.2	.035	11.7	do.	70.5
3	.720	65.0	.051	12.8	do.	66.9
4	.721	64.8	.050	12.1	do.	69.3
5	.717	65.9	.050	11.4	do.	70.8
6	.726	63.4	.043	10.2	do.	69.3
7	.726	63.4	.046	9.4	do.	69.9
8	.731	62.1	.015	11.4	do.	69.6
9	.725	63.7	.030	12.0	do.	70.0
10	.731	62.1	.055	10.3	do.	69.5
11	.711	67.6	.08	-	do.	61
12	.761	54.5	.08	-	do.	67
13	.756	55.7	.06	-	do.	70
14	.721	64.7	.03	-	do.	70
15	.746	58.1	.05	-	do.	72
Average	.728	62.9	.048	11.2	do.	69.0

Premium-price gasoline

1	0.708	68.4	0.040	11.2	Neg.	76.3
2	.723	64.2	.035	9.8	do.	75.7
3	.724	63.9	.055	10.8	do.	75.5
4	.726	63.4	.030	11.3	do.	75.6
5	.691	73.4	-	10.5	-	75.5
6	.723	64.2	.030	9.1	do.	76.1
7	.722	64.5	.070	11.5	do.	76.2
8	.732	61.9	-	9.6	-	76.5
Average	.719	65.3	.043	10.5	do.	75.9

Third-grade gasoline

1	0.727	63.1	0.04	6.5	Neg.	51.8
2	.733	61.5	.020	8.7	do.	58.3
3	.736	60.8	-	6.3	do.	47.8
4	.739	60.0	.04	7.0	do.	56.4
5	.745	58.4	.045	5.5	do.	52.3
6	.743	58.9	.02	7.0	do.	54.8
7	.704	69.5	.061	13.4	do.	61.4
8	.718	65.5	-	10.5	-	55.0
9	.727	63.0	.03	8.0	do.	58.0
10	.737	60.4	-	6.4	-	59.5
Average	.731	62.1	.026	7.9	Neg.	55.5

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

DETROIT, MICH.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
83	120	142	168	224	275	350	399	1.0	1.7	1
83	117	137	160	216	266	334	386	1.0	2.0	2
81	117	175	169	216	257	330	391	1.0	1.7	3
81	114	138	162	223	275	351	397	.9	2.0	4
86	117	137	161	215	268	345	396	1.0	1.0	5
90	128	157	185	237	284	353	400	.9	1.8	6
90	135	163	189	237	283	349	392	1.0	2.2	7
83	127	153	180	226	276	355	394	1.0	3.5	8
86	121	147	178	236	296	370	408	1.0	2.0	9
87	131	161	188	235	283	349	401	1.0	2.0	10
82	111	130	152	183	220	290	346	1	2	11
92	136	153	164	178	192	245	332	1	1	12
86	136	158	177	207	252	335	396	1	1	13
80	124	151	179	231	282	352	397	1	1	14
98	152	183	211	263	313	370	406	1	1	15
86	126	152	175	222	268	339	389	1.0	1.9	Average

Premium-price gasoline (continued)

87	121	137	155	192	229	292	359	.9	2.3	1
84	128	155	176	213	252	309	368	.9	2.3	2
83	119	143	171	227	285	357	407	1.0	2.1	3
87	123	149	116	230	284	349	394	.9	1.2	4
85	110	118	125	148	193	280	390	1.0	1.0	5
95	143	167	188	220	252	300	357	1.0	2.0	6
76	110	134	159	219	279	353	410	1.1	1.4	7
78	118	150	183	241	290	348	387	1.0	1.0	8
84	122	144	159	211	258	324	384	1.0	1.7	Average

Third-grade gasoline (continued)

95	143	174	202	247	292	347	390	1.0	2.0	1
98	148	177	201	242	283	341	388	1.0	1.0	2
101	158	192	215	254	295	352	398	1.0	1.3	3
99	150	183	211	257	302	368	428	.9	1.1	4
110	161	192	216	262	313	376	417	1.0	1.5	5
92	151	191	221	274	330	388	425	1.0	1.5	6
78	113	128	146	184	223	292	341	.8	3.7	7
89	126	147	170	226	278	364	403	1.0	1.5	8
90	138	163	186	227	270	340	390	1.0	2.0	9
86	134	178	208	258	306	369	419	1.0	2.5	10
94	142	173	198	243	289	354	400	1.0	1.8	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

NEW ORLEANS, LA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.730	62.3	0.054	9.4	Neg.	70.8
2	.730	62.3	.037	9.5	do.	69.5
3	.735	61.0	.048	8.9	do.	70.3
4	.738	60.2	.052	10.9	do.	69.8
5	.743	58.9	.058	9.2	do.	70.5
6	.731	62.2	.030	12.8	Neg.	69
7	.737	60.6	.044	10.7	do.	70.5
Average	.735	61.0	.046	10.2	do.	70.1

Premium-price gasoline

1	0.720	65.0	0.026	8.1	Neg.	77.3
2	.722	64.5	.017	8.3	do.	75.3
3	.722	64.5	-	7.5	do.	75
4	.723	64.1	.019	8.4	do.	75.5
5	.739	60.0	.073	9.5	do.	76
6	.746	58.2	.062	7.7	do.	76.0
Average	.729	62.6	.039	8.3	do.	75.8

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

NEW ORLEANS, LA.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
96	142	174	202	250	288	350	411	1.1	2.7	1
97	135	167	200	252	295	354	398	1.1	3.0	2
97	144	173	200	244	284	343	400	1.0	2.3	3
96	140	170	195	238	282	352	395	1.0	3.8	4
96	132	163	194	252	295	353	395	1.0	2.0	5
91	119	145	173	246	305	366	401	1.0	3.0	6
91	131	157	191	252	302	356	390	.9	2.1	7
95	135	164	194	248	293	353	399	1.0	2.7	Average

Premium-price gasoline (continued)

105	144	164	177	209	240	297	366	1.2	2.1	1
105	144	167	186	218	247	298	350	1.0	2.0	2
104	148	167	183	211	241	294	341	1.0	2.0	3
109	148	168	186	218	245	295	356	1.0	2.0	4
98	141	168	193	235	277	342	387	1.0	3.0	5
102	142	172	201	253	297	353	395	1.0	1.5	6
104	145	168	188	224	258	313	366	1.0	2.1	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

ST. LOUIS, MO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.707	68.6	0.082	12.2	Neg.	69.3
2	.719	65.3	.053	12.5	do.	69.4
3	.728	62.9	.029	10.6	do.	68.7
4	.724	63.9	.041	10.8	do.	69.7
5	.727	63.1	.049	10.3	do.	70.2
6	.728	62.9	.035	10.4	do.	70.2
7	.731	62.1	.051	10.0	do.	68.5
8	.732	61.8	.064	9.7	do.	70.5
9	.739	60.0	.061	10.3	do.	70.2
10	.744	58.7	.106	10.6	do.	69.7
Average	.728	62.9	.057	10.7	do.	69.6

Premium-price gasoline

1	0.704	69.5	-	11.0	Neg.	77.3
2	.708	68.4	-	8.6	do.	76.2
3	.709	68.1	-	9.4	do.	75.9
4	.711	67.6	-	6.20	-	76
5	.715	66.4	.022	10.4	-	75.1
6	.716	66.0	.017	10.4	-	76.0
7	.732	61.8	-	10.5	-	75.5
8	.730	62.3	-	9.8	-	76.8
9	.732	61.8	-	9.9	Neg.	75.6
10	.730	62.3	-	10.2	do.	75.3
11	.733	61.5	-	8.8	do.	74.6
12	.728	62.9	-	9.7	do.	75.4
Average	.721	64.8	-	9.6	do.	75.8

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

ST. LOUIS, MO.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
83	109	124	141	185	267	339	404	1.0	.9	1
85	117	142	169	220	273	343	390	.9	2.2	2
89	127	158	189	242	295	361	397	.9	2.5	3
90	123	148	174	229	285	355	406	.9	1.2	4
87	124	150	175	229	285	347	394	1.0	1.7	5
89	131	159	190	241	290	359	414	1.0	1.9	6
91	129	158	189	247	307	379	415	1.0	2.0	7
92	142	170	191	229	272	338	389	.9	1.4	8
85	126	153	181	242	308	381	416	1.0	1.6	9
86	126	158	191	245	299	364	405	1.0	2.4	10
87	125	152	179	231	288	357	403	1.0	1.8	Average

Premium-price gasoline (continued)

91	124	142	159	192	223	274	342	.9	1.5	1
99	122	133	146	183	239	337	406	1.0	1.0	2
96	120	131	144	180	237	335	401	1.0	1.3	3
120	146	154	162	182	210	277	365	.8	.7	4
89	118	139	165	223	269	314	367	1.0	1.3	5
93	126	151	176	223	268	331	383	1.0	1.7	6
90	131	160	187	230	276	342	383	1.0	2.2	7
89	129	157	182	232	280	353	409	1.0	1.2	8
90	129	155	180	230	281	349	396	1.0	1.3	9
92	135	163	187	228	272	334	358	1.0	1.5	10
93	129	161	191	244	293	355	395	1.0	1.7	11
91	133	164	194	254	313	355	417	1.0	1.8	12
94	129	151	173	217	263	321	385	1.0	1.4	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

ST. LOUIS, MO.

Third-grade gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.721	64.8	-	9.6	Neg.	57.0
2	.730	62.3	-	9.6	do.	50.6
3	.732	61.8	-	7.8	do.	51.0
4	.731	62.1	-	7.3	-	50.6
5	.731	62.1	-	8.5	Neg.	53.8
6	.736	60.8	-	6.2	do.	54.1
7	.735	61.0	-	9.3	do.	56.6
8	.740	59.7	-	7.6	do.	57.0
9	.745	58.4	-	9.5	do.	56.1
10	.749	57.4	-	4.3	do.	53.0
Average	.736	60.8	-	7.8	do.	53.9

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

ST. LOUIS, MO.

Third-grade gasoline (continued)

Distillation range, °F.										Item
I.B.P.	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
88	123	146	172	229	288	370	416	0.8	1.7	1
101	150	180	205	246	284	338	388	1.0	1.0	2
101	144	175	200	252	306	368	412	1.0	1.3	3
102	142	174	200	252	300	364	405	1.0	1.5	4
98	143	177	203	252	300	366	413	1.0	1.0	5
101	161	189	211	248	289	341	391	1.0	1.2	6
91	141	171	201	253	301	374	415	1.0	2.2	7
97	153	183	211	259	303	371	414	1.0	1.3	8
93	144	185	219	280	324	375	413	1.0	1.7	9
128	179	204	225	268	315	376	434	1.0	.8	10
100	148	178	205	254	301	364	410	1.0	1.3	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

MINNEAPOLIS-ST. PAUL, MINN.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.709	68.0	0.053	13.12	Neg.	70
2	.709	68.1	.039	12.65	do.	70
3	.718	65.7	-	13.15	-	70
4	.719	65.2	-	12.35	-	71.5
5	.720	65.0	.059	12.58	Neg.	69.3
6	.722	64.4	-	11.50	-	69.5
7	.723	64.1	.049	11.52	Neg.	70.3
8	.724	63.8	.043	11.12	do.	69.5
9	.727	63.1	.036	10.68	do.	69
10	.727	63.2	-	10.93	-	70
11	.727	63.1	-	-	-	69.2
Average	.722	64.5	.047	10.87	Neg.	69.8

Premium-price gasoline

1	0.683	75.8	0.034	11.66	Neg.	76.8
2	.699	71.0	.006	7.9	-	76.9
3	.725	63.6	.024	9.8	-	76.7
4	.706	68.8	-	12.55	-	78
5	.716	66.0	.030	11.0	Neg.	76
6	.718	65.7	-	11.40	-	77
7	.721	64.9	-	11.55	-	77
Average	.709	68.0	.024	10.69	-	76.9

Third-grade gasoline

1	0.730	62.4	-	9.35	-	50
2	.736	60.8	0.029	8.18	Neg.	50.8
3	.734	61.2	.025	8.0	do.	52
4	.735	61.1	-	9.18	-	57
5	.737	60.4	-	7.23	-	50
Average	.734	61.2	-	8.39	-	52

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

MINNEAPOLIS-ST. PAUL, MINN.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
83	108	129	147	202	264	357	415	1.0	2.7	1
84	110	127	146	195	257	356	414	1.0	2.1	2
83	119	145	173	225	277	346	392	.9	2.9	3
82	110	132	158	218	287	365	412	.9	2.6	4
81	111	134	161	218	276	352	393	.9	2.9	5
80	118	143	170	227	283	357	404	.8	2.5	6
83	122	151	180	235	284	355	391	.8	3.2	7
87	121	141	171	230	285	361	407	1.0	2.4	8
85	126	152	183	235	290	362	409	1.0	2.6	9
85	119	147	174	226	277	340	391	.8	2.3	10
93	127	156	189	250	311	387	464	1.5	3.5	11
84	117	141	168	224	281	358	408	1.0	2.7	Average

Premium-price gasoline (continued)

90	111	117	124	143	181	265	390	1.0	1.5	1
105	131	141	153	169	198	264	360	.8	1.3	2
91	124	148	175	228	281	349	392	.9	1.4	3
82	113	132	151	190	228	293	352	.9	3.1	4
87	120	140	165	228	269	320	365	1.2	1.8	5
84	122	147	172	218	265	335	382	1.0	2.2	6
84	114	136	161	217	276	357	420	1.0	2.0	7
89	119	137	157	199	243	312	380	1.0	1.9	Average

Third-grade gasoline (continued)

89	133	164	193	247	305	374	424	.9	2.0	1
95	147	179	209	260	313	379	426	.9	2.0	2
96	150	181	208	256	310	376	415	1.0	2.0	3
88	140	176	206	250	296	362	422	.9	2.3	4
95	150	183	210	257	307	369	413	.9	2.0	5
93	144	177	203	254	306	372	420	.9	2.1	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

OMAHA AND LINCOLN, NEBR.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper, strip test	ASTM octane number
	Specific	°API				
1	0.708	68.5	-	12.50	-	70
2	.715	66.5	-	12.12	-	70
3	.721	64.8	-	11.88	-	68
4	.722	64.4	-	11.05	-	68
5	.723	64.3	-	11.35	-	70
6	.724	64.0	-	11.25	-	69
7	.726	63.5	-	10.48	-	70
8	.727	63.2	-	10.90	-	-
9	.728	63.0	-	10.22	-	70
10	.729	62.6	-	10.00	-	70
11	.731	62.1	-	11.52	-	69
Average	.723	64.3	-	11.21	-	69.4

Premium-price gasoline

1	0.687	74.6	-	11.30	-	77
2	.698	71.3	-	8.55	-	77
3	.700	70.5	-	11.80	-	78
4	.715	66.5	-	-	-	74.0
5	.733	61.6	-	8.55	-	73
6	.740	59.7	-	8.05	-	76
7	.741	59.5	-	-	-	73.7
Average	.716	66.2	-	9.65	-	75.5

Third-grade gasoline

1	0.736	60.7	-	8.05	-	55
2	.737	60.4	-	6.90	-	50
3	.739	60.1	-	7.95	-	55
4	.739	60.1	-	6.18	-	50
Average	.738	60.3	-	7.27	-	52.5

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

OMAHA AND LINCOLN, NEBR.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
81	108	125	143	192	256	355	410	0.7	2.2	1
82	116	132	155	213	272	353	415	1.0	2.2	2
81	122	150	181	235	282	348	393	.7	3.6	3
85	122	146	174	233	294	369	410	.8	2.2	4
82	121	149	181	238	296	368	400	1.0	3.0	5
83	122	149	176	233	286	359	399	1.0	2.5	6
85	129	161	191	243	290	349	390	.9	2.5	7
84	130	160	189	242	291	360	420	.6	3.0	8
85	125	153	180	232	280	347	392	.8	2.0	9
87	130	160	187	241	292	358	393	.9	2.3	10
85	125	149	174	220	269	352	403	1.0	2.9	11
83	123	149	176	229	283	356	402	.9	2.6	Average

Premium-price gasoline (continued)

92	111	117	123	144	184	274	398	0.8	1.5	1
100	125	135	144	169	202	277	373	1.0	1.0	2
84	116	136	156	190	222	272	339	1.0	2.5	3
98	135	156	175	209	246	320	439	1.2	1.6	4
90	132	158	183	234	281	353	395	.9	1.6	5
91	146	179	207	253	295	353	387	1.1	1.4	6
97	145	176	205	253	302	359	403	1.0	1.2	7
93	130	151	170	207	247	317	391	1.0	1.5	Average

Third-grade gasoline (continued)

90	140	176	206	258	304	360	411	1.0	1.0	1
100	149	181	210	256	314	378	422	1.2	1.1	2
96	150	187	217	265	308	371	418	1.0	1.3	3
100	155	186	215	264	314	377	427	.8	.8	4
97	148	182	212	261	310	372	420	1.0	1.0	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

TULSA, OKLA.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.711	67.5	0.03	10.4	Neg.	70.1
2	.713	67.0	-	10.7	do.	70.0
3	.717	65.9	.06	12.0	do.	69.5
4	.723	64.2	.06	10.5	do.	69.1
5	.726	63.4	-	9.9	do.	70.3
6	.730	62.3	.09	10.8	do.	68.2
7	.728	62.9	-	9.4	do.	69.9
8	.727	63.1	.05	10.9	do.	69.3
9	.728	62.9	.03	10.5	do.	69.8
10	.731	62.1	.06	9.8	do.	69.7
11	.730	62.3	.05	10.0	do.	70.0
12	.732	61.8	.05	8.9	do.	69.9
13	.732	61.8	.03	9.2	do.	69.5
14	.732	61.8	-	8.6	do.	70.2
15	.732	61.8	-	8.9	do.	70.3
Average	.726	63.4	.051	10.0	do.	69.7

Premium-price gasoline

1	0.710	67.8	-	8.7	Neg.	74.6
2	.718	65.6	-	10.4	do.	77
3	.718	65.5	-	7.0	do.	74
4	.726	63.3	-	-	-	74.3
5	.727	63.2	-	11.9	-	78
6	.729	62.5	-	9.5	-	77
7	.732	61.8	0.06	9.4	Neg.	76.0
8	.732	61.8	.05	9.0	do.	75.9
9	.732	61.7	-	8.3	do.	75
10	.737	60.5	-	7.3	do.	76.2
Average	.726	63.4	-	8.2	do.	75.8

Third-grade gasoline

1	0.717	65.9	-	10.4	Neg.	56.0
2	.723	64.2	-	7.4	do.	50.5
3	.726	63.5	-	9.9	-	54
4	.735	61.0	-	7.5	Neg.	51.5
5	.742	59.2	.05	6.5	do.	54.2
6	.728	62.9	.05	8.9	do.	53.1
7	.739	59.9	-	7.5	do.	47.5
Average	.730	62.3	-	8.3	do.	52.4

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

TULSA, OKLA.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
86	114	133	154	202	265	356	406	1.0	1.9	1
87	121	146	171	225	270	341	392	1.0	2.3	2
80	114	138	164	219	274	346	397	1.1	2.5	3
84	116	142	169	223	278	348	395	1.0	2.3	4
84	121	154	185	240	290	356	390	1.0	2.3	5
88	124	149	173	217	267	343	402	1.0	2.3	6
87	122	149	176	231	290	367	413	.9	1.7	7
78	125	156	187	242	291	362	404	.8	2.5	8
85	131	164	190	247	295	364	413	1.1	3.0	9
87	132	162	192	246	295	356	394	.9	2.2	10
86	127	158	186	238	286	350	407	1.0	2.5	11
88	130	160	188	241	288	351	395	.9	2.4	12
90	132	165	196	252	307	366	400	1.0	2.7	13
86	131	165	194	243	286	352	391	1.0	3.3	14
88	131	162	190	241	289	356	399	1.0	2.0	15
86	125	154	181	234	285	354	400	1.0	2.4	Average

Premium-price gasoline (continued)

90	121	142	160	205	258	330	387	1.0	1.5	1
92	128	151	174	218	262	320	387	1.0	1.8	2
90	125	148	165	202	249	326	394	1.0	1.5	3
93.2	126	147	169	221	284	372	408	1.0	1.0	4
81	121	152	182	238	288	358	420	1.2	2.8	5
90	130	157	184	232	281	346	413	1.1	1.6	6
89	128	155	182	231	280	344	392	.9	1.9	7
88	124	149	176	235	296	362	404	1.0	1.5	8
96	134	163	187	242	297	355	402	1.0	1.5	9
96	139	169	200	250	301	360	402	1.0	1.3	10
91	128	153	178	227	279	347	401	1.0	1.5	Average

Third-grade gasoline (continued)

92	123	144	167	220	287	369	419	1.0	1.0	1
90	126	153	175	215	257	320	390	1.0	2.0	2
84	126	159	185	239	288	362	422	.9	2.1	3
95	144	177	205	255	303	371	415	1.1	1.4	4
98	154	190	214	261	310	373	428	1.0	1.4	5
92	132	164	192	245	298	373	418	1.1	1.5	6
92	140	178	214	280	337	389	422	1.0	1.5	7
92	135	166	193	245	297	365	416	1.0	1.6	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

HOUSTON, TEX.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.732	61.9	0.130	7.1	Neg.	67.8
2	.732	61.7	.030	6.7	do.	69.0
3	.733	61.4	.048	6.0	do.	70.2
4	.742	59.3	.052	9.0	do.	70.0
5	.733	61.6	.042	6.7	do.	69.7
6	.734	61.3	.039	6.0	do.	70.4
7	.743	58.9	.054	7.0	do.	69.6
8	.742	59.3	.030	5.8	do.	68.5
9	.745	58.4	.059	7.5	do.	69.8
10	.757	55.5	.037	6.5	do.	69.0
Average	.739	60.0	.052	6.8	do.	69.4

Premium-price gasoline

1	0.724	63.9	0.030	4.8	Neg.	78.5
2	.725	63.6	.031	4.3	do.	74.0
3	.730	62.4	.020	6.3	do.	75.8
4	.730	62.4	.020	5.7	do.	74.0
5	.736	60.8	.038	8.3	do.	69.9
6	.744	58.6	.029	4.0	do.	74.5
7	.737	60.5	.051	5.0	do.	74.2
8	.747	57.8	.049	5.3	do.	74.1
9	.760	54.7	.029	7.5	do.	73.5
Average	.737	60.5	.033	5.7	do.	74.2

Third-grade gasoline

1	0.731	62.1	0.02	7.6	Neg.	58.0
2	.733	61.5	.06	5.7	do.	60.3
3	.734	61.3	.02	4.7	do.	58.0
4	.738	60.2	.036	5.5	do.	59.2
5	.742	59.2	.039	7.0	do.	58.0
6	.742	59.2	.04	5.0	do.	57.5
7	.743	58.8	.051	4.8	do.	58.0
8	.744	58.6	.03	6.7	do.	57.5
9	.744	58.6	.049	6.7	do.	59.4
10	.752	56.6	.051	5.4	do.	57.0
Average	.740	59.7	.04	5.9	do.	58.3

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

HOUSTON, TEX.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
95	136	158	180	223	261	326	386	1.0	1.0	1
93	141	169	198	247	295	352	387	1.0	1.0	2
91	138	169	195	238	290	340	384	1.0	1.0	3
97	142	171	199	250	297	360	399	1.0	1.0	4
89	136	166	194	247	293	364	395	.5	1.0	5
100	146	175	201	253	295	349	400	.5	1.0	6
93	145	176	209	257	300	348	399	1.0	1.0	7
97	147	178	205	247	295	348	396	1.0	1.0	8
94	140	169	200	253	295	345	416	1.0	1.0	9
97	147	176	207	263	316	369	408	.5	1.0	10
95	142	171	199	248	294	350	395	1.0	1.0	Average

Premium-price gasoline (continued)

107	152	170	184	209	235	278	352	1.0	1.0	1
104	152	173	190	218	244	289	352	.5	1.0	2
93	143	169	194	233	274	332	377	.5	1.0	3
97	145	171	193	232	270	327	372	.5	1.0	4
97	143	175	203	248	296	353	420	.5	1.0	5
110	163	189	210	249	284	340	390	.5	1.0	6
106	148	171	193	234	284	345	386	.5	1.0	7
98	149	175	202	251	300	351	388	.5	1.0	8
92	138	167	200	258	314	372	408	1.0	1.0	9
100	148	173	197	237	278	332	383	.6	1.0	Average

Third-grade gasoline (continued)

91	147	176	200	240	279	330	384	.5	1.0	1
95	152	176	196	233	269	330	396	.5	1.0	2
100	151	181	205	256	300	352	396	1.0	1.0	3
101	155	183	205	247	290	349	412	1.0	1.0	4
94	149	179	207	249	295	396	511	1.0	1.5	5
101	156	188	211	250	288	344	408	.5	1.0	6
106	161	189	222	264	298	343	397	1.0	1.0	7
93	143	174	199	246	295	356	398	.5	1.0	8
91	138	171	212	271	306	350	398	1.0	1.0	9
98	156	187	211	257	316	385	433	.5	1.0	10
97	151	180	207	251	322	354	413	.8	1.1	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

DENVER, COLO.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.715	66.4	0.015	11.3	Neg.	70.2
2	.714	66.7	.023	12.2	do.	70.6
3	.720	65.0	.029	11.2	do.	69.3
4	.723	64.2	-	11.6	-	70
5	.724	64.0	-	11.9	-	69
6	.724	64.0	-	11.6	-	68
7	.727	63.1	-	9.9	-	70
8	.731	62.1	.029	9.4	Neg.	61.4
9	.736	60.8	.028	10.1	do.	69.0
10	.736	60.8	.026	9.4	do.	70.0
11	.740	59.7	.022	11.8	do.	70.0
12	.739	60.0	.014	8.7	do.	70.2
13	.738	60.2	-	8.0	do.	70
Average	.728	62.9	.023	10.5	do.	69.2

Premium-price gasoline

1	0.711	67.5	-	10.1	-	76.0
2	.716	66.1	-	10.95	-	77
3	.727	63.1	-	6.7	-	74.0
4	.732	61.8	-	9.1	-	75.8
5	.740	59.8	-	6.4	-	76.0
6	.743	58.9	-	4.4	-	69.1
7	.743	58.9	-	8.5	-	76.8
Average	.730	62.3	-	8.0	-	75.0

Third-grade gasoline

1	0.724	63.9	0.024	10.0	Neg.	56.3
2	.729	62.6	.032	9.7	do.	53.4
3	.727	63.1	.029	8.5	do.	44.4
4	.732	61.9	-	8.7	-	33.5
5	.741	59.5	.016	7.9	Neg.	56.4
6	.747	57.9	.032	7.1	do.	57.8
7	.746	58.2	.023	7.7	do.	57.4
8	.742	59.2	.019	7.4	do.	55.1
9	.751	56.9	.12	6.4	do.	52.0
Average	.738	60.2	.037	8.2	do.	51.8

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

DENVER, COLO.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
84	114	136	154	208	267	362	404	1.0	2.7	1
79	109	130	155	213	273	359	411	1.1	3.5	2
84	114	138	164	219	276	353	394	1.0	3.3	3
81	117	142	171	230	290	364	404	1.0	2.6	4
81	119	149	182	240	290	357	397	0.8	3.2	5
83	124	153	184	237	288	356	394	1.0	3.0	6
86	133	161	187	232	270	324	376	1.0	2.0	7
80	125	160	191	244	293	356	386	1.0	5.0	8
87	130	164	197	253	309	378	407	1.0	3.4	9
85	136	170	197	245	294	361	398	1.0	2.7	10
89	138	170	197	247	295	388	402	1.0	2.7	11
91	139	173	203	254	300	363	397	1.0	2.9	12
100	138	171	200	253	302	371	407	1.0	2.0	13
85	126	155	183	237	288	361	398	1.0	3.0	Average

Premium-price gasoline (continued)

90	125	145	165	201	235	290	354	1.0	1.6	1
84	120	143	166	214	267	340	410	1.0	2.0	2
108	140	158	174	211	255	340	402	1.1	1.1	3
98	136	163	191	243	292	354	394	1.0	2.1	4
109	146	175	202	250	294	347	398	1.0	1.0	5
120	164	189	210	250	293	351	404	1.0	.7	6
93	141	172	201	253	301	364	406	1.0	1.7	7
100	139	164	187	232	276	341	395	1.0	1.6	Average

Third-grade gasoline (continued)

86	128	158	185	228	274	341	386	1.0	3.0	1
91	134	165	194	245	303	376	411	1.0	2.9	2
86	132	162	190	242	299	376	417	1.0	3.0	3
97	144	176	206	250	300	367	408	1.0	2.8	4
94	145	180	209	256	303	367	405	1.0	2.6	5
95	154	193	222	267	309	368	409	1.1	2.0	6
92	145	184	229	273	320	377	415	1.0	2.3	7
96	153	191	220	268	314	374	407	1.0	1.4	8
97	159	195	222	271	326	394	428	1.0	2.0	9
93	144	178	209	256	305	371	410	1.0	2.6	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

SALT LAKE CITY, UTAH.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.733	61.6	0.05	7.6	Neg.	68.0
2	.737	60.5	.028	11.9	do.	70.6
3	.739	60.0	.026	9.6	do.	70.4
4	.739	60.1	.069	8.3	do.	69.8
5	.739	60.0	.025	11.1	do.	70.5
6	.744	58.6	.052	9.3	do.	70.4
7	.763	54.0	.062	7.6	do.	68.1
8	.738	60.2	.029	9.5	do.	70.1
Average	.741	59.5	.043	9.4	do.	69.7

Third-grade gasoline

1	0.744	58.6	0.021	9.1	Neg.	62.2
2	.745	58.4	.016	7.8	do.	58.7
3	.746	58.1	.019	10.0	do.	61.4
4	.748	57.7	.017	8.2	do.	62.2
5	.748	57.7	.027	9.1	do.	65.9
6	.749	57.3	.016	9.2	do.	62.7
7	.749	57.3	.035	9.8	do.	62.4
8	.751	56.8	.131	7.6	do.	59.1
9	.752	56.7	.031	8.0	do.	61.2
Average	.748	57.7	.035	8.8	do.	61.8

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

SALT LAKE CITY, UTAH.

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
92	124	151	179	231	281	358	406	1.0	2.3	1
86	121	155	191	256	314	379	402	1.0	5.0	2
91	131	164	194	248	297	365	401	1.0	3.8	3
80	126	162	194	246	292	354	384	1.0	5.0	4
79	122	155	177	248	300	366	406	1.0	4.0	5
80	126	160	194	250	306	368	400	1.0	4.5	6
93	136	162	182	215	275	388	428	1.0	4 3	7
83	125	156	184	240	295	364	392	1.0	4.5	8
86	126	158	189	241	295	368	402	1.0	4.2	Average

Third-grade gasoline (continued)

80	132	179	216	282	330	399	414	1.0	5.5	1
86	134	169	200	252	305	380	412	1.0	4.0	2
84	135	180	216	278	331	395	419	1.0	5.5	3
81	140	182	216	276	330	394	418	1.0	5.5	4
84	134	172	204	256	308	380	402	1.0	4.0	5
85	136	178	214	276	324	386	416	1.0	4.5	6
89	140	169	192	240	308	398	422	1.0	5.5	7
84	142	185	217	269	315	376	400	1.0	3.5	8
85	137	180	216	276	330	392	430	1.0	4.0	9
84	137	177	210	267	320	389	415	1.0	4.7	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

SEATTLE, WASH.

Regular-price gasoline

Item	Sample taken, 1935	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
		Specific	°API				
A	November	0.740	59.7	0.23	11.1	Neg.	69
B	do.	.747	57.8	.03	9.7	do.	69
C	do.	.745	58.4	.27	9.4	do.	68
D	do.	.748	57.6	.25	11.1	do.	68
E	do.	.736	60.8	.21	8.8	do.	68
F	do.	.738	60.2	.08	10	do.	70
G	do.	.729	62.5	.10	11	do.	68.5
Average	do.	.740	59.6	.17	10.2	do.	68.6

Premium-price gasoline

A	November	0.726	63.4	0.02	6.6	Neg.	81
B	do.	.744	58.7	.03	9.2	do.	81
C	do.	.742	59.2	.03	9.4	do.	80.5
D	do.	.742	59.3	.03	7.5	do.	80
E	do.	.731	62	.03	8.1	do.	80
F	do.	.738	60.1	.08	8.4	do.	80
G	do.	.738	60.2	.06	7.9	do.	79.5
Average	do.	.737	60.4	.04	8.2	do.	80.3

Third-grade gasoline

A	November	0.750	57.2	0.12	7.7	Neg.	61
B	do.	.756	55.7	.05	7.9	do.	63
C	do.	.745	58.4	.21	8.2	do.	60
D	do.	.752	56.6	.16	8.2	do.	63
E	do.	.747	58.0	.13	8.2	do.	62
F	do.	.749	57.3	.20	7.2	do.	60
G	do.	.745	58.3	.13	7.8	do.	61
Average	do.	.749	57.4	.14	7.9	do.	61.4

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

SEATTLE, WASH.

Regular-price gasoline (continued)

I.B.P.	Distillation range, °F. Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Resid.	Loss	
88	131	165	200	255	304	362	402	1	2	A
92	140	174	205	253	301	362	405	1	2	B
88	138	170	205	255	307	364	401	1	2	C
91	143	173	210	264	316	378	409	1	2.5	D
94	135	160	187	242	294	363	414	1	1.5	E
91	135	163	193	240	289	353	396	1	2	F
92	124	146	174	224	277	352	409	1	1.5	G
91	135	164	196	248	298	362	405	1	1.9	Average

Premium-price gasoline (continued)

101	146	164	181	209	238	284	342	1	1	A
93	145	176	206	253	301	357	402	1	2	B
95	142	175	206	254	300	357	402	1	1.5	C
96	153	179	205	239	275	329	384	1	1	D
94	134	156	177	223	262	314	372	1	1	E
100	143	168	195	237	280	338	385	1	1	F
101	143	166	190	227	268	339	408	1	1	G
97	144	169	194	235	275	331	385	1	1.2	Average

Third-grade gasoline (continued)

102	152	186	214	262	314	370	415	1	2	A
93	146	188	225	276	325	385	421	1	1.5	B
88	142	175	210	263	308	380	428	1	1.5	C
98	158	196	224	270	316	374	410	1	2	D
97	144	175	205	260	318	383	422	1	1.5	E
101	157	189	219	265	308	376	428	1	1	F
95	140	170	205	257	308	394	420	1	1.5	G
96	148	183	215	265	314	380	421	1	1.6	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

LOS ANGELES, CALIF.

Regular-price gasoline

Item	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
1	0.730	62.2	0.28	-	Neg.	67.0
2	.735	61.1	.12	-	do.	70.0
3	.736	60.9	.14	-	do.	70.0
4	.737	60.4	.28	-	do.	71.0
5	.738	60.1	.07	-	do.	71.0
6	.738	60.1	.30	-	do.	71.0
7	.740	59.7	.11	-	do.	69.0
8	.743	58.8	.15	-	do.	69.0
9	.744	58.7	.02	-	do.	69.5
10	.744	58.7	.25	-	do.	70.0
11	.744	58.6	.02	-	do.	70.0
12	.746	58.2	.23	-	do.	69.0
13	.750	57.2	.13	-	do.	67.0
Average	.740	59.6	.16	-	do.	69.5

Premium-price gasoline

1	0.729	62.7	0.02	-	Neg.	80.0
2	.729	62.6	.05	-	do.	81.0
3	.729	62.5	.02	-	do.	80.0
4	.732	61.8	.06	-	do.	80.0
5	.732	61.7	.05	-	do.	79.5
6	.733	61.5	.04	-	do.	79.5
7	.733	61.4	.04	-	do.	81.0
8	.736	60.8	.05	-	do.	80.5
9	.746	58.2	.02	-	do.	80.0
10	.746	58.1	.08	-	do.	77.0
Average	.735	61.1	.04	-	do.	79.9

Third-grade gasoline

1	0.748	57.6	0.10	-	Neg.	63.0
2	.748	57.6	.13	-	do.	63.0
3	.757	55.5	.04	-	do.	63.0
4	.752	56.7	.03	-	do.	63.0
5	.748	57.6	.09	-	do.	63.0
6	.738	60.2	.28	-	do.	66.0
7	.742	59.1	.04	-	do.	63.0
8	.751	56.8	.07	-	do.	63.0
9	.743	58.8	.06	-	do.	64.0
10	.748	57.6	.10	-	do.	64.0
11	.749	57.4	.03	-	do.	64.0
12	.753	56.3	.07	-	do.	62.0
13	.757	55.4	.10	-	do.	62.0
Average	.749	57.4	.09	-	do.	63.3

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

LOS ANGELES, CALIF.

Regular-price gasoline (continued)

Distillation range, °F.										Item
I.B.P.	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
82	135	163	188	224	269	342	396	1.0	3.0	1
86	132	171	194	240	282	339	390	1.1	1.9	2
98	144	170	193	246	280	343	400	.9	2.1	3
90	128	156	182	234	282	356	414	1.1	1.9	4
88	140	168	194	233	279	350	414	1.1	1.7	5
89	145	176	204	247	290	348	401	1.1	1.9	6
86	136	171	203	256	307	363	398	1.1	2.5	7
88	136	169	200	246	292	360	404	1.0	1.7	8
89	140	170	202	250	291	351	394	1.1	2.5	9
82	131	169	202	257	303	365	402	1.1	2.1	10
84	140	174	209	254	296	357	394	1.2	2.5	11
84	137	177	208	259	303	363	400	1.0	2.0	12
98	150	186	218	266	310	365	403	1.0	1.5	13
88	138	171	200	247	291	354	401	1.1	2.1	Average

Premium-price gasoline (continued)

100	150	170	186	210	235	279	340	0.9	1.1	1
89	136	154	178	220	260	304	362	.9	1.1	2
104	149	171	187	212	238	285	356	.9	1.1	3
97	139	161	184	227	267	320	378	.9	1.5	4
94	138	162	184	222	256	320	380	1.1	1.9	5
92	134	164	190	232	270	330	380	.9	1.9	6
92	135	160	188	234	277	332	382	.8	2.0	7
94	144	170	190	226	266	340	412	.8	3.2	8
92	144	180	210	252	294	348	396	1.0	2.3	9
93	149	183	224	260	298	360	402	1.0	1.7	10
95	142	168	192	230	266	322	379	.9	1.8	Average

Third-grade gasoline (continued)

87	142	182	210	262	306	378	420	1.0	2.0	1
90	140	181	217	274	316	366	430	1.2	2.2	2
96	156	197	230	278	321	381	416	1.0	2.0	3
86	139	180	219	272	316	381	418	1.1	1.9	4
86	139	180	219	258	307	370	418	.9	1.9	5
91	138	168	198	255	299	374	424	1.2	1.2	6
86	136	170	201	249	296	368	423	1.0	2.6	7
93	151	191	221	268	316	383	420	1.2	1.8	8
91	143	178	211	261	307	371	416	.8	2.2	9
90	144	178	212	267	318	383	418	1.0	2.0	10
84	136	178	217	267	325	384	414	1.0	2.3	11
92	150	200	233	276	327	384	418	.9	1.9	12
98	164	205	233	276	318	378	414	.8	1.2	13
90	144	184	217	266	313	377	419	1.0	1.9	Average

TABLE 1.- Motor-gasoline survey, winter, 1935-36
(Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF.

Regular-price gasoline

Item	Sample taken, 1935-36	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
		Specific	°API				
A	February	0.740	59.8	0.18	9.6	Neg.	70.5
B	do.	.746	58.1	.036	9.9	do.	70.5
C	do.	.745	58.3	.034	9.7	do.	70.5
D	do.	.737	60.4	.058	-	do.	65.5
E	do.	.736	60.6	.19	10.4	do.	71
F	do.	.734	61.3	.15	-	do.	70.5
G	do.	.738	60.2	.20	-	do.	70.5
H	do.	.747	57.8	.20	10.5	do.	71.5
I	do.	.734	61.2	.13	8.6	do.	70.5
K	do.	.735	61.1	.15	9.0	do.	70.5
L	do.	.737	60.5	.10	7.8	do.	70.5
M	do.	.734	61.3	.10	10.4	do.	70.5
Average	do.	.739	60.0	.127	9.6	do.	70.2

Premium-price gasoline

A	February	0.726	63.5	0.020	6.5	Neg.	80.5
B	do.	.743	58.9	.032	9.0	do.	79
C	do.	.743	58.9	.042	7.5	do.	79
D	do.	.733	61.4	.053	-	do.	80.5
E	do.	.726	63.3	.037	5.7	do.	80.5
F	do.	.737	60.4	.035	9.4	do.	79
G	do.	.729	62.6	.037	-	do.	81
H	do.	.729	62.7	.033	8.1	do.	81
I	do.	.736	60.8	.084	9.0	do.	79.5
K	do.	.733	61.5	.043	8.1	do.	80.5
Average	do.	.733	61.4	.042	7.9	do.	80.0

Third-grade gasoline

A	February	0.744	56.6	0.094	8.0	Neg.	60
B	do.	.750	57.2	.074	-	do.	62
C	do.	.754	56.1	.096	6.7	do.	59.5
D	do.	.755	56.0	.083	-	do.	60
E	do.	.747	58.0	.15	7.5	do.	63.5
F	do.	.743	58.9	.047	-	do.	62
G	do.	.745	58.4	.23	-	do.	64
H	do.	.750	57.2	.12	9.4	do.	64
I	do.	.744	58.7	.069	-	do.	62
K	do.	.745	58.4	.070	8.3	do.	62
L	do.	.743	59.0	.22	8.5	do.	65.5
M	do.	.746	58.2	.13	8.8	do.	62.5
Average	do.	.748	57.7	.115	8.2	do.	62.3

TABLE 1.- Motor-gasoline survey, winter, 1935-36 - Continued
(Data for samples from different cities)

SAN FRANCISCO BAY REGION, CALIF.

Regular-price gasoline (continued)

Distillation range, °F.									
I.B.P.	Percent recovered						End point	Percent		Item
	10	20	30	50	70	90		Recov.	Loss	
86	130	168	-	262	-	362	393	96	-	A
94	142	176	-	254	-	354	400	96	-	B
90	137	172	-	251	-	360	402	96.5	-	C
92	133	164	-	245	-	365	405	96.5	-	D
96	128	161	-	251	-	361	400	95	-	E
92	135	164	-	238	-	332	384	97	-	F
87	129	161	-	256	-	362	396	96	-	G
87	128	165	-	260	-	359	389	96.5	-	H
92	134	162	-	238	-	338	398	97	-	I
95	137	166	-	238	-	343	383	96.5	-	K
95	138	168	-	244	-	344	388	96.5	-	L
92	136	162	-	237	-	355	406	96	-	M
91	134	166	-	248	-	353	395	96.3	-	Average

Premium-price gasoline (continued)

107	149	165	-	205	-	276	335	98	-	A
92	140	174	-	248	-	350	396	97	-	B
99	147	170	-	225	-	329	386	97.5	-	C
100	147	168	-	220	-	329	406	97.5	-	D
105	149	165	-	208	-	286	333	97.5	-	E
100	145	175	-	234	-	321	382	96	-	F
102	135	155	-	221	-	313	374	97.5	-	G
98	137	152	-	220	-	313	370	98	-	H
104	140	168	-	240	-	344	383	96	-	I
104	142	164	-	220	-	329	405	97	-	K
101	143	166	-	224	-	319	377	97.2	-	Average

Third-grade gasoline (continued)

95	155	197	-	277	-	366	400	97	-	A
93	150	188	-	267	-	382	419	96.5	-	B
98	163	202	-	277	-	375	413	97	-	C
100	159	197	-	276	-	377	415	97	-	D
93	145	181	-	267	-	375	418	97	-	E
90	139	173	-	259	-	385	425	96	-	F
90	138	181	-	274	-	368	392	95	-	G
92	157	198	-	274	-	380	412	95	-	H
93	143	177	-	267	-	384	428	96	-	I
92	139	175	-	269	-	380	428	97	-	K
95	143	173	-	257	-	375	422	97	-	L
100	144	183	-	263	-	374	422	97	-	M
94	148	185	-	269	-	377	416	96.4	-	Average

TABLE 2.- Motor-gasoline survey, winter, 1935-36
(Average values for samples from different cities)

Regular-price gasoline

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
Boston	0.733	61.5	0.07	9.9	Neg.	69.3
New York	.734	61.3	.067	10.01	do.	69.7
Washington 1/	.735	61.0	.067	9.8	do.	69.2
Pittsburgh	.731	62.1	.061	10.0	do.	70.0
Cleveland	.731	62.1	.06	10.5	do.	69.9
Chicago	.723	64.2	.043	11.5	do.	69.7
Detroit	.728	62.9	.048	11.2	do.	69.0
New Orleans	.735	61.0	.046	10.2	do.	70.1
St. Louis	.728	62.9	.057	10.7	do.	69.6
Minneapolis-St. Paul	.722	64.5	.047	10.87	do.	69.8
Omaha 2/	.723	64.3	-	11.21	do.	69.4
Tulsa	.726	63.4	.051	10.0	do.	69.7
Houston	.739	60.0	.052	6.8	do.	69.4
Denver	.728	62.9	.023	10.5	do.	69.2
Salt Lake City	.741	59.5	.043	9.4	do.	69.7
Seattle	.740	59.6	.17	10.2	do.	68.6
Los Angeles	.740	59.6	.16	-	do.	69.5
San Francisco	.739	60.0	.127	9.6	do.	70.2
Average	.732	61.8	.07	10.1	do.	69.6

1/ Includes samples taken in Baltimore.
2/ Includes samples taken in Lincoln, Nebr.

TABLE 2.- Motor-gasoline survey, winter, 1935-36 - Continued
(Average values for samples from different cities)

Regular-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		City
	10	20	30	50	70	90		Resid.	Loss	
90	135	164	194	239	281	341	394	1.8	1.6	Boston
89	132	162	190	242	285	347	396	1.4	1.8	New York
89	132	161	191	241	286	349	395	1.0	1.8	Washington 1/
89	136	164	193	242	284	347	395	1.0	2.0	Pittsburgh
88	131	161	189	238	285	356	405	1.0	2.6	Cleveland
86	119	146	173	229	283	356	401	1.0	2.8	Chicago
86	126	152	175	222	268	339	389	1.0	1.9	Detroit
95	135	164	194	248	293	353	399	1.0	2.7	New Orleans
87	125	152	179	231	288	357	403	1.0	1.8	St. Louis
84	117	141	168	224	281	358	408	1.0	2.7	Minneapolis-St. Paul
83	123	149	176	229	283	356	402	.9	2.6	Omaha 2/
86	125	154	181	234	285	354	400	1.0	2.4	Tulsa
95	142	171	199	248	294	350	395	1.0	1.0	Houston
85	126	155	183	237	288	361	398	1.0	3.0	Denver
86	126	158	189	241	295	368	402	1.0	4.2	Salt Lake City
91	135	164	196	248	298	362	405	1	1.9	Seattle
88	138	171	200	247	291	354	401	1.1	2.1	Los Angeles
91	134	166	-	248	-	353	395	-	-	San Francisco
89	130	159	186	238	286	353	399	1.1	2.3	Average

1/ Includes samples taken in Baltimore.
2/ Includes samples taken in Lincoln, Nebr.

TABLE 3.- Motor-gasoline survey, winter, 1935-36
(Average values for samples from different cities)

Premium-price gasoline

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper-strip test	ASTM octane number
	Specific	°API				
Boston	0.739	60.0	0.05	9.3	Neg.	76.6
New York	.730	62.3	.06	9.8	do.	77.6
Washington 1/	.747	57.9	.069	8.9	do.	76.4
Pittsburgh	.732	61.8	.042	9.5	do.	76.6
Cleveland	.723	64.2	.024	9.7	do.	76.2
Chicago	.725	63.7	.04	10.0	do.	75.5
Detroit	.719	65.3	.043	10.5	do.	75.9
New Orleans	.729	62.6	.039	8.3	do.	75.8
St. Louis	.721	64.8	-	9.6	do.	75.8
Minneapolis-St. Paul	.709	68.0	.024	10.69	-	76.9
Omaha 2/	.716	66.2	-	9.65	Neg.	75.5
Tulsa	.726	63.4	-	8.2	do.	75.8
Houston	.737	60.5	.033	5.7	do.	74.2
Denver	.730	62.3	-	8.0	-	75.0
Salt Lake City	No premium-price gasolines included					
Seattle	.737	60.4	.04	8.2	Neg.	80.3
Los Angeles	.735	61.1	.04	-	do.	79.9
San Francisco	.733	61.4	.042	7.9	do.	80.0
Average	.729	62.7	.042	9.0	do.	76.7

1/ Includes samples taken in Baltimore.
2/ Includes samples taken in Lincoln, Nebr.

TABLE 3.- Motor-gasoline survey, winter, 1935-36 - Continued
(Average values for samples from different cities)

Premium-price gasoline (continued)

Distillation range, °F.										
I.B.P.	Percent recovered						End point	Percent		City
	10	20	30	50	70	90		Resid.	Loss	
96	139	160	180	213	247	308	367	1.0	1.6	Boston
93	136	163	187	230	271	331	386	1.2	1.7	New York
97	143	167	188	226	268	337	390	1.1	1.5	Washington 1/
94	137	162	184	224	262	321	375	.9	1.6	Pittsburgh
91	134	162	187	228	266	328	385	1.0	2.3	Cleveland
92	128	152	177	225	274	342	400	1.0	2.0	Chicago
84	122	144	159	211	258	324	384	1.0	1.7	Detroit
104	145	168	188	224	258	313	366	1.0	2.1	New Orleans
94	129	151	173	217	263	321	385	1.0	1.4	St. Louis
89	119	137	157	199	243	312	380	1.0	1.9	Minneapolis St. Paul
93	130	151	170	207	247	317	391	1.0	1.5	Omaha 2/
91	128	153	178	227	279	347	401	1.0	1.6	Tulsa
100	148	173	197	237	278	332	383	.6	1.0	Houston
100	139	164	187	232	276	341	395	1.0	1.6	Denver
	No premium-price gasolines included									Salt Lake City
97	144	169	194	235	275	331	385	1	1.2	Seattle
95	142	168	192	230	266	322	379	.9	1.8	Los Angeles
101	143	166	-	224	-	319	377	-	-	San Francisco
95	136	159	181	223	264	326	384	1.0	1.7	Average

1/ Includes samples collected in Baltimore.
2/ Includes samples collected in Lincoln, Nebr.

TABLE 4.- Motor-gasoline survey, winter, 1935-36
(Average values for samples from different cities)

Third-grade gasolines

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Copper, strip test	ASTM octane number
	Specific	°API				
Boston	0.738	60.2	-	9.3	Neg.	62.9
New York	.741	59.4	-	8.8	-	65.6
Washington 1/	.742	59.2	.095	8.9	Neg.	65.6
Pittsburgh	No third-grade gasolines included					
Cleveland	.733	61.5	.038	6.6	do.	50.6
Chicago	.734	61.3	.032	8.5	do.	56.6
Detroit	.731	62.1	.026	7.9	do.	55.5
New Orleans	No third-grade gasolines included					
St. Louis	.736	60.8	-	7.8	Neg.	53.9
Minneapolis-St. Paul	.734	61.2	-	8.39	-	52.
Omaha 2/	.738	60.3	-	7.27	-	52.5
Tulsa	.730	62.3	-	8.3	Neg	52.4
Houston	.740	59.7	.04	5.9	do.	58.3
Denver	.738	60.2	.037	8.2	Neg.	51.8
Salt Lake City	.748	57.7	.035	8.8	do.	61.8
Seattle	.749	57.4	.14	7.9	Neg.	61.4
Los Angeles	.749	57.4	.09	-	do.	63.3
San Francisco	.748	57.7	.115	8.2	do.	62.3
Average	.739	60.0	.065	8.1	do.	57.9

1/ Includes samples collected in Baltimore.
2/ Includes samples collected in Lincoln, Nebr.

TABLE 4.- Motor-gasoline survey, winter, 1935-36 - Continued
(Average values for samples from different cities)

Third-grade gasolines - (Continued)

Distillation range, °F.										City
I.B.P.	Percent recovered						End point	Percent		
	10	20	30	50	70	90		Resid.	Loss	
90	139	170	197	243	284	343	400	2.0	1.4	Boston
89	139	172	202	252	296	351	406	.8	2.1	New York
85	129	158	188	243	288	347	398	1.1	1.8	Washington 1/
			No third-grade gasolines included							Pittsburgh
104	159	188	213	252	279	353	400	1.0	1.7	Cleveland
99	147	177	204	250	296	357	412	1.0	1.6	Chicago
94	142	173	198	243	289	354	400	1.0	1.8	Detroit
			No third-grade gasolines included							New Orleans
100	148	178	205	254	301	364	410	1.0	1.3	St. Louis
93	144	177	203	254	306	372	420	.9	2.1	Minneapolis-St. Paul
97	148	182	212	261	310	372	420	1.0	1.0	Omaha 2/
92	135	166	193	245	297	365	416	1.0	1.6	Tulsa
97	151	180	207	251	322	354	413	.8	1.1	Houston
93	144	178	209	256	305	371	410	1.0	2.6	Denver
84	137	177	210	267	320	389	415	1.0	4.7	Salt Lake City
96	148	183	215	265	314	380	421	1	1.6	Seattle
90	144	184	217	266	313	377	419	1.0	1.9	Los Angeles
94	148	185	-	269	-	377	416	-	-	San Francisco
94	144	177	205	254	301	364	411	1.0	1.9	Average

1/ Includes samples collected in Baltimore.
2/ Includes samples collected in Lincoln, Nebr.

TABLE 5.- Motor-gasoline survey, March 1931[1/]
(Average values of samples from different cities)

Competitive-price gasolines[2/]

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Octane number	Distillation range, °F.					
	Specific	°API				I.B.P.	Percent recovered				End point
							10	20	50	90	
Boston	0.744	58.7	0.07	9.1	64	88	139	175	257	362	398
New York	.741	59.5	.07	9.7	64	94	144	174	256	365	401
Washington	.745	58.4	.09	8.9	68	89	138	175	263	364	397
Pittsburgh	.747	57.9	.07	9.5	68	98	141	174	258	368	402
Atlanta	.744	58.7	.06	8.1	65	90	139	174	265	371	407
Cleveland	.738	60.2	.07	10.2	65	90	138	172	258	371	400
Chicago	.737	60.5	.07	10.0	63	86	132	169	263	381	413
New Orleans	.745	58.4	.08	8.9	66	90	138	173	265	368	400
St. Louis	.736	60.8	.08	9.1	56	94	138	169	254	376	409
Minneapolis-St. Paul	.733	61.5	.06	9.7	59	91	131	163	249	374	407
Omaha	.733	61.5	.06	9.0	54	93	139	171	252	370	407
Tulsa	.734	61.3	.02	9.4	60	90	141	170	250	366	406
Dallas	.741	59.5	.05	8.2	63	90	146	179	260	369	407
Denver	.738	60.2	.05	8.6	59	82	128	158	242	366	400
El Paso	.744	58.7	.11	7.9	60	90	142	178	256	353	384
Salt Lake City	.747	57.9	.03	9.4	64	82	134	171	259	369	398
Seattle	.745	58.4	.10	9.3	66	92	139	177	262	371	409
Los Angeles	.750	57.2	.11	8.4	62	92	143	181	264	381	413
San Francisco	.750	57.2	.09	9.1	62	90	138	176	262	383	413
Average	.738	60.2	.07	9.1	63	90	138	173	258	371	404

1/ Data compiled from table 3, Rept. of Investigations 3129; table 1, Rept. of Investigations 3142, and table 8, Rept. of Investigations 3175 (see footnotes 10 and 4).

2/ Includes third-grade gasolines.

TABLE 6.- Motor-gasoline survey, March 1931[1/]
(Average values of samples from different cities)

Premium-price gasolines

City	Gravity		Sulphur, %	Reid vapor pressure, lb.	Octane number	Distillation range, °F.					End point
	Specific	°API				I.B.P.	Percent recovered				
							10	20	50	90	
Boston	0.741	59.5	0.08	9.1	80	92	142	175	252	354	389
New York	.741	59.5	.07	9.0	76	92	142	169	251	363	402
Washington	.757	55.4	.10	8.7	80	92	138	167	240	357	396
Pittsburgh	.744	58.7	.07	8.6	79	95	139	169	243	357	399
Atlanta	.741	59.5	.08	7.9	77	90	144	175	250	349	389
Cleveland	.742	59.2	.05	8.8	78	96	147	173	229	341	386
Chicago	.735	61.0	.09	10.2	69	88	129	161	243	361	395
New Orleans	.744	58.7	.08	7.4	74	97	148	180	260	358	393
St. Louis	.733	61.5	.05	9.2	74	95	135	167	247	364	402
Minneapolis-St. Paul	.733	61.5	.04	8.7	73	92	134	161	234	356	405
Omaha	.729	62.6	.07	10.2	73	93	130	156	239	362	395
Tulsa	.729	62.6	.04	9.9	76	89	132	157	233	359	399
Dallas	.743	58.9	.09	8.0	74	92	141	172	249	364	403
Denver	No premium-price motor fuels sampled										
El Paso	.743	58.9	.09	6.1	74	88	147	176	244	334	368
Salt Lake City	.759	54.9	.07	6.9	78	91	143	172	243	356	388
Seattle	.748	57.7	.09	7.6	86	96	143	176	251	354	397
Los Angeles	.750	57.2	.09	8.1	81	93	139	173	250	351	389
San Francisco	.752	56.7	.08	8.1	83	94	139	172	249	351	395
Average	.743	58.9	.07	8.5	77	92	139	170	245	355	395

1/ Data compiled from table 3, Rept. of Investigations 3129; table 1, Rept. of Investigations 3142; and table 8, Rept. of Investigations 3175 (see footnotes 10 and 4).

TABLE 7.- Comparison of sulphur content, 1931-36
(Averages for samples from different cities)

City	Average sulphur content, percent S.						
	Competitive price			Premium price			Third grade
	Winter 1935-36	1/Aug. 1931	1/Mar. 1931	Winter 1935-36	Aug. 1931	Mar. 1931	Winter 1935-36
Boston	0.07	0.07	0.07	0.05	0.07	0.08	-
New York	.067	.07	.07	.06	.06	.07	-
Washington	2/.067	.09	.09	2/.069	.09	.10	2/.095
Pittsburgh	.061	.07	.07	.042	.07	.07	-
Cleveland	.06	.06	.07	.024	.07	.05	.038
Chicago	.043	.08	.07	.04	.08	.09	.032
New Orleans	.046	.07	.08	.039	.06	.08	-
St. Louis	.057	.07	.08	-	.04	.05	-
Minneapolis-St. Paul	.047	.04	.06	.024	.04	.04	-
Omaha	-	.06	.06	-	.07	.07	-
Tulsa	.051	.02	.02	-	.03	.04	-
Denver	.023	.03	.05	-	.04	-	.037
Salt Lake City	.043	.04	.03	-	.04	3/.07	.035
Seattle	.17	.10	.10	.04	.08	.09	.14
Los Angeles	.16	.11	.11	.04	.09	.09	.09
San Francisco	.127	.09	.09	.042	.07	.08	.115

1/ Includes third-grade gasolines.
2/ Includes samples collected in Baltimore.
3/ Average of two samples.

TABLE 8.- Comparison of Reid Vapor Pressures, 1931-36
(Averages for samples from different cities)

City	Average Reid vapor pressures, pounds						
	Competitive price			Premium price			Third grade
	Winter 1935-36	1/Aug. 1931	1/Mar. 1931	Winter 1935-36	Aug. 1931	Mar. 1931	Winter 1935-36
Boston	9.9	8.2	9.1	9.3	7.9	9.1	9.3
New York	10.01	8.0	9.7	9.8	7.5	9.0	8.8
Washington	2/9.8	7.6	8.9	2/8.9	7.6	8.7	2/8.9
Pittsburgh	10.0	8.2	9.5	9.5	7.6	8.6	-
Cleveland	10.5	7.8	10.2	9.7	6.9	8.8	6.6
Chicago	11.5	8.1	10.0	10.0	7.8	10.3	8.5
New Orleans	10.2	7.7	8.9	8.3	6.6	7.4	-
St. Louis	10.7	7.4	9.1	9.6	7.4	9.3	7.8
Minneapolis-St. Paul	10.87	7.9	9.7	10.69	7.9	8.7	8.39
Omaha	3/11.21	7.3	9.0	3/9.65	7.6	10.1	3/7.27
Tulsa	10.0	7.3	9.4	8.2	7.8	9.9	8.3
Denver	10.5	6.8	8.6	8.0	7.3	-	8.2
Salt Lake City	9.4	7.3	9.4	-	7.0	4/6.9	8.8
Seattle	10.2	8.2	9.3	8.2	7.3	7.6	7.9
Los Angeles	-	7.3	8.4	-	7.4	8.1	-
San Francisco	9.6	8.3	9.1	7.9	7.6	8.1	8.2

1/ Includes third-grade gasolines.
2/ Includes samples collected in Baltimore.
3/ Includes samples collected in Lincoln, Nebr.
4/ Average of two samples.

TABLE 9.- Comparison of octane numbers, 1931-36
(Averages for samples from different cities)

City	Average octane numbers						
	Competitive price			Premium price			Third grade
	Winter 1935-36	1/Aug. 1931	1/Mar. 1931	Winter 1935-36	Aug. 1931	Mar. 1931	Winter 1935-36
Boston	69.3	65	64	76.6	75	80	62.9
New York	69.7	65	64	77.6	73	76	65.6
Washington	2/ 69.2	68	68	2/ 76.4	76	80	2/65.6
Pittsburgh	70.0	67	68	76.6	77	79	-
Cleveland	69.9	60	65	76.2	74	78	50.6
Chicago	69.7	59	63	75.5	74	69	56.6
New Orleans	70.1	60	66	75.8	76	74	-
St. Louis	69.6	55	56	75.8	76	74	53.9
Minneapolis-St. Paul	69.8	58	59	76.9	72	73	52
Omaha	3/69.4	60	54	3/75.5	70	73	3/52.5
Tulsa	69.7	60	60	75.8	73	76	52.4
Denver	69.2	55	59	75.0	69	-	51.8
Salt Lake City	69.7	58	64	-	77	78	61.8
Seattle	68.6	65	66	80.3	83	86	61.4
Los Angeles	69.5	63	62	79.9	81	81	63.3
San Francisco	70.2	64	62	80.0	82	83	62.3

1/ Includes third-grade gasolines.
2/ Includes samples collected in Baltimore.
3/ Includes samples collected in Lincoln, Nebr.

TABLE 10.- Motor-gasoline survey, winter, 1935-36

(Summary of data of the survey)

Test	Regular-price gasolines			Premium-price gasolines			Third-grade gasolines		
	Min.	Range of 90%	Max.	Min.	Range of 90%	Max.	Min.	Range of 90%	Max.
Specific gravity)	0.763	0.747-0.715	0.707	0.779	0.749-0.702	0.684	0.757	0.753-0.724	0.704
Degrees A.P.I.)	54.0	58.0-66.5	68.6	50.1	57.5-70.0	75.4	55.4	56.5-64.0	69.5
Sulphur content, %	.014	.02-.15	.30	.006	.02-.08	.179	.016	.02-.20	.23
Reid vapor press.,lb.	5.8	7.1-12.4	13.15	3.1	5.0-11.5	12.55	4.2	5.5-10.4	13.4
Copper-strip test		Neg.			Neg.			Neg.	
ASTM octane number	61	68-71	72	69.1	73.7-80.5	81	33.5	50-66	69.1
Initial boiling point	79	82-98	100	76	84-107	128	78	84-104	128
10 percent recovered	108	114-144	155	110	115-150	178	113	127-161	179
20 percent recovered	124	132-175	186	117	134-176	201	128	150-196	205
30 percent recovered	141	156-206	218	124	147-209	224	146	183-224	233
50 percent recovered	178	207-255	266	143	180-252	261	184	230-276	282
70 percent recovered	192	267-308	316	181	225-301	314	223	270-326	337
90 percent recovered	245	335-377	388	264	273-359	378	292	330-393	399
End point	332	385-415	464	331	343-415	439	341	388-430	511
Dist. loss	.9	1-3.5	5.0	.5	1-3	3.6	.5	1-3.5	5.5

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

22.09
R32mb
Cop2

R. I. 3312

SEPTEMBER 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

THE IMPROVED PERMISSIBLE FLAME SAFETY LAMPS

BY

L. C. ILSLEY, A. B. HOOKER, AND E. J. COGGESHALL

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3312
September 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

THE IMPROVED PERMISSIBLE FLAME SAFETY LAMPS[1]

By L.C. Ilsley[2], A.B. Hooker[3] and E.J. Coggeshall[4]

GENERAL

The service given by a flame safety lamp is twofold: It provides a safe light and it gives evidence of the condition of the mine atmosphere relative to the presence of methane and the deficiency of oxygen. Since the development of portable electric mine lamps, flame safety lamps are little used for lighting, but their use as detectors is as extensive and important as ever.

A flame safety lamp in the hands of a trained user will definitely show methane concentrations as low as 1 percent, which is well below the lower explosive limit of methane and air. However, this does not always suffice, especially when making studies of mine ventilation, where it may be desireable to detect concentrations below 1/2 of 1 percent.

The usual method of detecting methane with a flame safety lamp is as follows: The lamp is normally carried with a luminous flame of 1/2 to 1-3/8 inches in height. When the user observes an elongation of the flame that denotes the presence of methane, the percentage of the gas may be roughly estimated by the amount of the elongation; or, if a closer estimate is desired, the luminous flame is lowered until only a trace of yellow remains and the height of the gas cap can be observed.

In low concentrations of methane gas caps are difficult to see, and changes in the height or elongation of the visible flame are very small. Furthermore, both gas cap and high flame are affected by several factors other than the methane concentration. Thus, even where the present types of permissible flame safety lamps are considered adequate, detection of methane in a mine may be sure, or just a guess; or, the gas may be missed altogether, depending upon the user, the lamp and the conditions under which the test is made.

---

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3312."

2 Electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 Associate electrical engineer, U.S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

4 Senior electrical engineering aide, U.S. Bureau of Mines, Pittsburgh Experiment Station.

Observation of gas caps can be made only when in the dark and after the eye has become adjusted to the new conditions. This procedure has unsafe features in that a lamp with lowered flame for capping is more easily extinguished than the hotter lamp with high flame. Then, too, during the interval of inspection the observer is without illumination of the roof and surroundings from which hazards may develop. Thus, while there is an apparent need for better ways of observing gas caps, there is even greater need of a means of detecting gas while in an illuminated place. An investigation was made along these lines by the Electrical Section of the Bureau.

PRELIMINARY INVESTIGATION

A preliminary investigation was made to determine better ways of observing gas caps; also, to develop a practical way of observing and measuring flame elongation.

At first, the gas caps were observed by means of peephole and slot attachments through which the top of the gas cap was lined up with the corresponding image of the cap on the lamp glass. The peephole was mounted on a slider and was moved vertically to line up with the changed height of the cap, the amount of movement showing on a suitable scale.

Other tests were made with a small platinum exploring wire that could be moved vertically and turned horizontally into the gas cap and by its color or temperature show the physical limits of the cap and therefore any change in height.

The third device consisted of a glass bead supported on a fixed platinum wire in the gas cap which, in a fused state, colors the cap and makes it more visible. The amount of rise was observed by a peephole or movable slot device. None of these methods is considered practicable because the use of a peephole-type device, with uncolored cap, is limited to the few people who can readily see a gas cap; it is thus not generally applicable. The use of the platinum wire and the glass bead require an operating rod through the bowl or fount, also a suitable scale.

Studies of the capping flames and gas caps showed the following facts relative to the ease and accuracy of observing them:

Above the capping flame is a fuel cap of nearly colorless flame. So few people can see the fuel cap that it is usually not mentioned in publications relative to flame safety lamps. When a lamp is taken into a mixture of methane* appears above the capping flame and fuel cap. This, too, is so nearly colorless that only the lower parts and edges of the cap are visible to most observers until in 1 to 1-1/2 percent methane, when the cap is said to be complete.

The usual procedure in testing with capping flame underground requires that the cap be observed as soon as possible after lowering the traveling flame, before the lamp cools appreciably. This is because the cap is larger and seen more easily when the lamp is hot.

* and air, an additional cap of burning methane

The lamp cools so quickly immediately after lowering the flame that if the observer delays to permit adjustment of his eyes or to change his position there will be an appreciable difference in the heigh t of cap and resultant error in estimating the percentage.

Approximate heights of caps for round-wick permissible flame safety lamps (Koehler and Wolf), with 70° naphtha fuel, are given in table 1. The values were obtained by means of the platinum exploring wire and scale attachment after the lamp temperature and capping flame were stable.

Table 1

Flame or cap	Total height above wick tube, inch	Increase in total height, inch
Original capping flame, with just a trace of yellow	1/8 inch	--
Fuel cap	3/16 inch	1/16 inch
1/4 percent methane cap	7/32 inch	1/32 inch
1/2 percent methane cap	1/4 inch	1/32 inch
1 percent methane cap	5/16 inch	1/16 inch
1-12 percent methane cap	3/8 inch	1/16 inch

The first column shows the total heights of the caps above the wick tube; the second column shows the rate of increase in height of cap with increase of methane. The very small changes in cap height indicate the difficulty of estimating the lower percentages of methane, even by the few observers who can see such caps distinctly.

Observation of High Flames

Changes in the heights of the high or luminous flame were observed b y means of parallel lines etched around the glass. These lines also serve as a means of leveling the lamp while observing.

Another method of measuring small changes in the height of luminous flames was to mount a thermometer with its bulb just above the flame, yet with the working range of its scale outside the lamp and visible to the observer. An all-metal thermometer was also used, similarly mounted but filled with carbon tetrachloride, the expansion and contraction of which were indicated as a change in pressure on a suitable gage.

The investigation showed the unstability of the flames and the importance of using a definite height of flame when testing.

Flames regularly used in flame safety lamps are 1/2 to 1-3/8 inches high. The elongation of the 1-3/8- inch flame is about three times as great as that of the 1/2-inch flame in the same percentage of methane, due mostly to the difference in the size of the flame. It is due partly to the higher temperature and greater speed of ventilation in the lamp with the higher flame.

From this it would seem that the higher the testing flame, the greater the indication. However, there is a maximum practicable limit as to flame height, determined by the ventilation of the lamp and the range of methane concentration in which it is to be used. Assuming that the chief use of the lamp will be in 0 to 2 percent methane, the maximum height of flame, as set in fresh air, should be that which, in a 2-percent mixture, is about to redden and smoke for lack of ventilation. This height of flame was used in practically all later tests. With proper fuel such a flame has a well defined tip suitable for observation by means of a lined glass.

Bimetal Devices

Although thermometers are not suitable for measuring changes in flame height, because of their fragility, cost, and weight, their use led to the use of thermostatic bimetal devices that had none of their disadvantages and were just as accurate. The investigation then consisted chiefly of developing and applying an adequate bimetal device.

The first bimetal units were used with the capping flame and were mounted so that they could be turned out of position when the traveling flame was used. Later, only luminous flames were used, and the bimetal unit was mounted permanently above the flame.

Numerous bimetal coils were tested, including combinations of the following shapes and dimensions: Conical spiral with pointer at center; flat spiral with pointer at center; flat spiral with pointer at outer end; bimetal strip 0.010 and 0.005 inch thick; 1/8 inch, 3/16 inch, and 1/4 inch wide; and 8, 10, and 12 inches long.

The most suitable of the preliminary units was the spiral coil of 0.005-inch bimetal, 1/4-inch wide, mounted at its center to a spider at the top of the glass. The movement of the free outer end of the spiral was indicated by its shadow on a circular scale on the glass. However, normal movement of this unit was adversely affected by uneven heating due to the presentation of different parts of the spiral to the flame as the spiral unwound and to side movement or waving of the flame as the lamp was tilted.

Better performance in this respect was obtained by making the bimetal unit in the form of a helix and mounting it higher above the flame in the inner gauze. To obtain an additional steadying effect, the helix was mounted in a conical metal tube which acts as a draft tube, guiding the hot burned gases up past the helix without direct interference and cooling by the incoming gases.

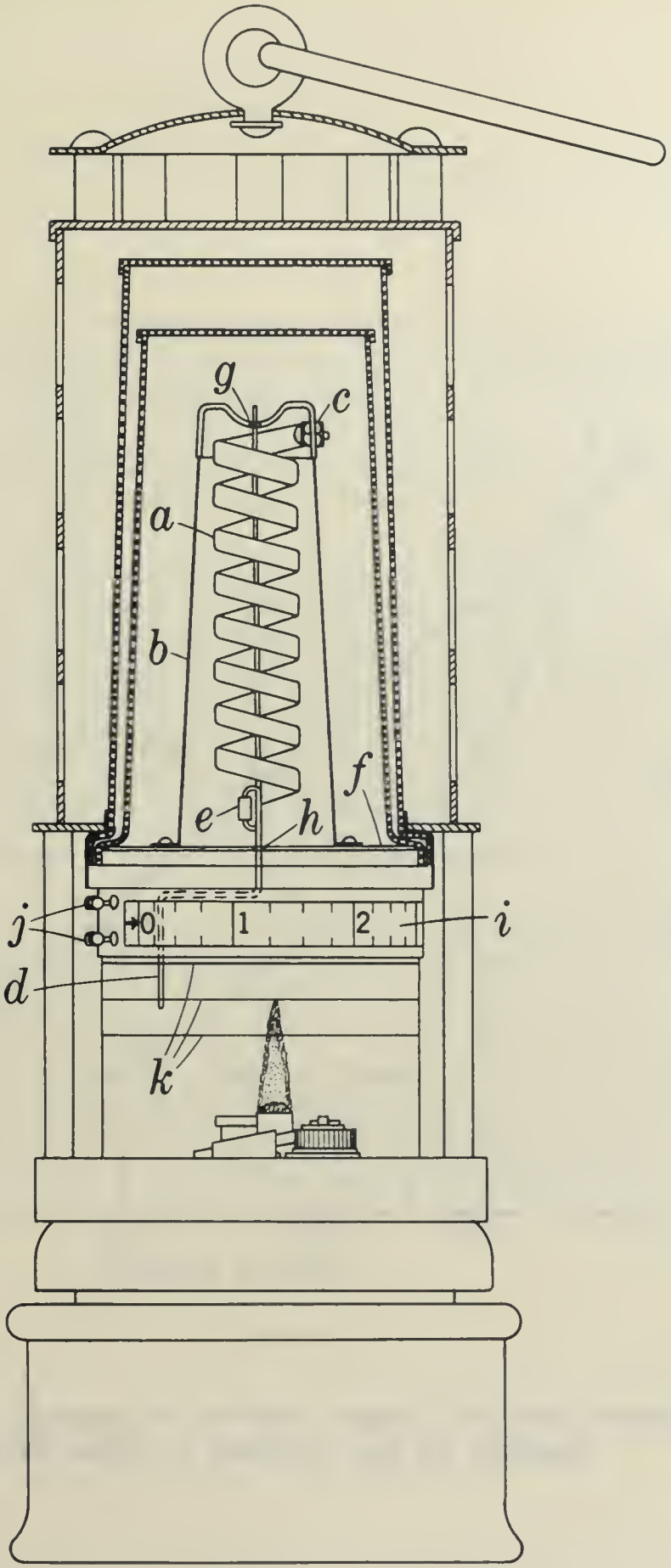


Figure 1.— Section showing assembly of the bimetal unit in an improved flame safety lamp.

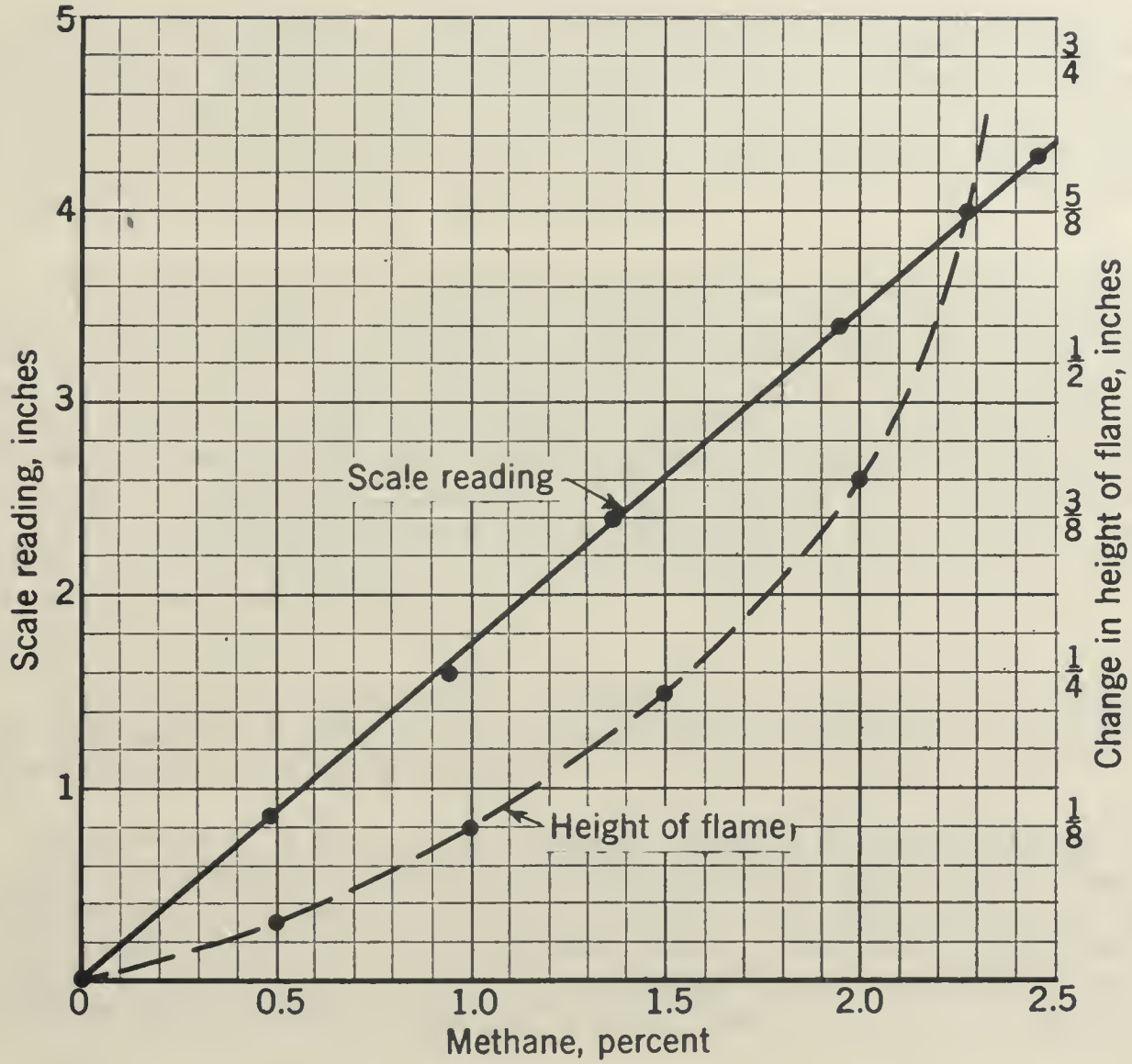


Figure 2.— Relative changes of the flame height and scale reading of the improved lamps in methane and air mixtures.

The Bimetal Helix

The preliminary tests having shown that the helix-type device was preferable, further investigation centered about perfecting it and determining what modifications in the design and usage of the permissible lamps were desirable relative to its operation.

The several types of helix devices tried out included various lengths of bimetal, also two different helixes — the unwind or expanding type having the high expansion material inside and its reverse, the wind-up or contracting type. Several different methods of attaching or mounting the helix unit as well as different kinds and shapes of conical tubes surrounding the helix, were used. The modifications in the design and usage of the lamps were all based upon the desire for greater stability of the lamp flame.

The final model of the helix unit with its scale and lined glass assembled in a lamp is shown in figure 1. The helix unit a, made of bimetal 0.005 inch thick, 3/16 inch wide and approximately 15 inches long, is mounted in a stainless steel tube b, being attached at its top end to the yoke at c and at its bottom end to pointer rod d at e. The tube itself is supported by the spider f, which is mounted between the glass and the gauzes. As mounted in the lamp, the pointer rod d is suspended by the helix, being attached at the bottom and free to move a little vertically through guide holes g and h as the helix elongates or contracts, and to turn freely as the helix coils or uncoils, yet restrained as to movement from the vertical.

The fastening of the helix at point c is purposely left loose to permit free alinement of the helix, thus preventing side thrust on the pointer and excessive friction at points g and h.

The scale i is made of translucent material through which the shadow of the pointer is clearly visible. This increases the east of observing the position of the pointer, especially when it is back of a lamp post. The scale marking is in inches deflection of the bimetal device. The scale is held on the glass by one or more coiled springs j, yet may be moved at will to permit zero scale setting or better observation of a high flame.

Three parallel lines k spaced 1/4 inch apart, are etched either on the inside or outside of the glass. The center line indicates the height of flame that gives approximately the best results with the bimetal unit. The upper and lower lines serve as gages of the rise and lowering of the flame, respectively.

Performance of Bimetal Helix Device

Typical performance curves of improved lamps, obtained in gas and normal air mixtures, are represented in figure 2. A comparison of these curves shows the great difference in relative ease of observing the change in flame height and scale reading. In 1/4-percent methane the flame rises only about 1/64 inch, whereas the increase in scale reading is nearly 1/2 inch. Even with 1 percent gas the flame rise is only 1/8 inch as against over 1-1/2 inches increase in scale reading.

Thus, the increment of change of flame height is small at the low concentrations in which most observations would be made and becomes greater in the higher concentrations where observation is of less importance, while the bimetal device actuated by this flame gives an approximately straight-line scale deflection of 1-1/2 to 2 inches for each percent of methane.

THE PRODUCTION MODELS

The manufacturers of the permissible types of flame safety lamps cooperated throughout the investigation. As soon as a satisfactory design of the bimetal helix device was available, each manufacturer prepared to apply it to his lamps. Each submitted drawings and samples of the device as he expected to produce it and requested extension of the original approvals to cover its application to his permissible lamps.

To distinguish the new lamps from the regular permissible types and from possible future designs, they are designated as "Koehler or Wolf Improved Flame Safety Lamp, Bureau of Mines Model."

Besides the bimetal device with its scale and lined glass, the production models of the improved lamps embody several suggested changes in design as follows:

The Wick

The bimetal device is applied to round-wick lamps only. The flat-wick lamps of both companies are off center, that is, not concentric with the lamp bowl. This, together with the unsymmetrical shape of the wick, gives a variable and unsatisfactory operation of the device, for it is possible to have a different position of the flame with respect to the bimetal unit each time the lamp is assembled.

The round wicks of the improved lamps are larger in diameter than the regular wicks. Sometimes, wicks fit so loosely in the wick holder that bumping the lamp may cause downward movement of the wick and lowering of the flame. The wicks of the improved lamps fit sufficiently tight to prevent such slipping, and, fitting snugly in the wick tub, permit closer adjustment of the flame height.

Height of Flame

The temperature of a lamp largely determines the speed of the gases through the lamp and thus its sensitiveness or quickness of response when brought into a gassy mixture. The temperature of the lamp increases with the height of flame; a high flame is therefore desirable.

The flames recommended for the improved lamps (15/16 inch for the Koehler lamp and 13/16 inch for the Wolf lamp when adjusted to the middle line of the glass) are approximately the maximum flames that give straight line deflection of the bimetal device through the 0- to 2- percent range of methane in air.

Fuel

A fuel is normally designated by its gravity, together with its initial and end distillation points. Two fuels may have the same gravity yet be very different as to their distillation temperatures and behavior in the lamp. One may have volatile fractions that distill at relatively low temperatures, while the other may be composed wholly of fractions that distill at the intermediate temperatures.

Fuel distillation at the lower temperatures is unduly affected by changes in the temperature of the lamp and causes unsteady flame. Fuel that distills only at the higher temperatures may leave gummy deposits in the wick and cotton packing that interfere with the normal flow of fuel to and through the wick.

The fuel for the improved lamps should be 68°-70° naphtha of medium distillation range. Naphtha has a narrower distillation range than gasoline and is therefore more suitable. None of the regular motor gasolines is a satisfactory lamp fuel.

Vented Bowl

Unsteady flame may result from increased pressure within the bowl, as the bowl increases in temperature, causing forced feed through the wick and wick assembly. To minimize this effect, the fount of each improved lamp is vented through the wick-adjustment tube. Burning tests with lamps whose vents could be opened or closed at will showed definitely that the vented lamps burned less fuel per hour, besides burning with a steadier flame.

Assembly and Servicing

The improved lamps are designed especially for detecting methane underground. Particular attention should therefore be paid to correct assembling and servicing. Openings in the wick assembly, such as at the top of the wick adjustment screw, due to the caps not being in place, permit leakage of fuel vapor other than through the wick and intermittent ignition of the leaking fuel that causes a pulsating flame. There should be no end-play in the wick adjustor that permits accidental movement of the wick and change of flame height without turning the adjustment screw.

The lamp bowl should be uniformly filled with the cotton. The wick should be long enough to be trailed out through the cotton in intimate contact with the fuel supply. After filling, all excess fuel should be removed by shaking the inverted lamp bowl.

The glass of the Wolf lamp has the word "TOP" etched on it to indicate its correct assembly in the lamp; the Koehler glass, unmarked in this respect, should be similarly assembled with the parallel lines toward the top. In this position, the bottom of the scale coincides with the third line; the fourth line is covered.

APPROVALS

The improved lamps were approved in December 1935 as extensions of the original approvals of the Koehler and Wolf round-wick lamps. The approval numbers are as follows: Koehler lamps - steel frame, No. 201-A; aluminum frame, No. 203-A, Wolf lamps - brass frame, No. 204; aluminum frame, No. 205.

In addition to tests relative to their performance in detecting methane, the lamps were subjected to the following safety tests:

Tests of the improved lamps were made in moving mixtures u p to 2,500 feet per minute velocity to determine the effect of the bimetal unit upon safety. Tests were made with and without the bimetal unit and the corresponding temperatures of the inner gauze determined by means of thermocouples. The tests were made under the conditions of maximum heating, namely, air flow vertically down with respect to the lamp; air velocity, 2,500 feet per minute; natural gas concentration, 8.6 percent.

A thermocouple was attached at the center of the gauze top and at the center of the hottest side; that is, the lamp was turned until maximum side temperature was attained.

In no test did the inner gauze reach 800°C., which is well below the fusion temperation of the metal.

With the bimetal device the top of the gauze was hotter than the side; without the device, the side was hotter than the top. Maximum temperatures occurred without the device, thus in every test the bimetal device tended to distribute the heat, causing more uniform heating of the gauzes and a lower maximum temperature.

The lined glasses were tested for safety by dropping a 1-pound lead weight from a height of 4 feet, also subjecting them to a temperature test in which the glasses were heated to a temperature of 212°F., then sprayed with water at approximately 60°F.

Operation of the Improved Lamps

When the improved lamp is lighted, the pointer of the bimetal unit makes approximately two complete turns before coming to rest with the flame adjusted to the middle line of the glass.

With flame adjustment as needed, let the lamp reach approximately constant temperature while going into the mine. Before going beyond fresh air, reset the flame height with tip of flame on the middle line. Either move the scale until the pointer shadow is at zero, or note where the pointer is on the scale.

Proceed to place where test for methane is to be made. If methane is present, the lamp will give a double indication, (a) by an increase in the height of flame above the line of setting; and (b) by a movement of the pointer as

shown by the shadow on the scale. In this observation, first note the change in height of flame, and then the resultant scale reading.

The pointer moves steadily, then slows and drifts. The point at which drift begins is approximately the true inches deflection of the device. The drift is due to the gradual change in temperature of the lamp from the increased flame and should be ignored.

Unless going directly from a higher to a much lower concentration, it is not necessary that the scale reading be at zero before taking the lamp into another concentration. The improved lamp with its bimetal device gives a definite and reliable indication of the presence of methane. Thus one should not put too much emphasis on zero scale setting, but depend more upon the observed changes in height of flame and scale reading.

Tests in Atmospheres Deficient in Oxygen

When a flame safety lamp is brought into a mixture that is deficient in oxygen, the flame rises -- a phenomenon sometimes called "search for oxygen," then changes color and lowers. If no methane is present, the resultant flame height may be above, equal to, or below the original, depending upon the relative amounts of methane and oxygen present.

The bimetal device of the improved lamps, being actuated by the heat of the flame, responds positively to the initial flame rise and, in general, gives a scale reading proportional to the total height of the flame, irrespective of its color or visibility. The improved lamp therefore definitely responds to the methane by its scale reading while showing the approximate degree of oxygen deficiency by the appearance of its flame.

CONCLUSIONS

Although changes in design have tended to make the improved lamp more stable as to burning, the accuracy of the lamp as a detector is limited by variations in the flame setting, such as may result from appreciable changes in air movements and temperature. The improved lamp is therefore not a precision device, yet when taken into any methane concentration that can be detected with the present-type lamps its response is unmistakable and can be seen by anyone.

Detection with the improved lamps is independent of illumination, making possible the use of auxiliary lighting for inspection purposes wherever permitted.

The use of the improved lamps should add years of usefulness to fire bosses and others who are no longer able to see gas caps.

The definite flame height assures a lamp of nearly constant temperature - one that is always sensitive to the presence of methane and less liable to be extinguished than a lamp that is adjusted from high to low flame for special testing.

The bimetal device has mechanical strength comparable to that of the gauzes and adds very little to the weight or cost of the lamp. It is readily removed when cleaning the lamp. Its position in the lamp is such that the user may ignore it if he so desires, using his high flame and capping flames as formerly.

The authors believe that these lamps will fill a distinct need in gassy mines provided they are used only as detectors, noting the presence of methane by the positive change in scale reading and flame height without undue emphasis as to the percentage of gas present. Continued use of the lamps will undoubtedly increase individual skill and confidence in particular lamps but never to the point where they will compete with or replace the more expensive and accurate portable methane detectors of the indicating type.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

72.09
32mo
Cop2

R. I. 3314 October 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

FLOTATION OF VERMONT TALC-MAGNESITE ORES

BY

J. BRUCE CLEMMER AND S. R. B. COOKE

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R. I. 3314
October 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

FLOTATION OF VERMONT TALC-MAGNESITE ORES [1]

By J. Bruce Clemmer[2] and S. R. B. Cooke [3]

INTRODUCTION

In the experimental concentration of Vermont talc-magnesite ores, talc concentrates and tailings rich in magnesite were obtained. On account of the possible market value of the magnesite, the tailings are given consideration in this report along with the talc.

After some comments about the talc industry, the magnesite that accrued as tailings will be considered. Then the flotation results will follow.

ACKNOWLEDGMENT

This work was carried on under the supervision of W. H. Coghill, supervising engineer, Ore-Dressing Section, Metallurgical Division.

TALC

Talc is one of the most widely used natural minerals. It is a highly useful material and finds application in over 60 diversified industries. It is usually marketed in powder form and finds its most important use as a filler or loading material in the paper, rubber, soap, and paint industries. It is a useful ingredient in cement and plaster, and considerable amounts are used in the production of roofing materials and linoleums. It is also used as a dusting agent in processing many commodities. The demands of the cosmetic trade are largely supplied by imports. Although cosmetic talc commands a premium, the annual consumption is relatively unimportant compared to the bulk trade. Talc or soapstone marketed in the massive form as cut blanks of various shapes is important to several industries. The reports of Ladoo[4] and MacConkey[5] give more complete information on the uses of talc and the grades demanded by the consumers.

1/ This paper presents the results of work done under a cooperative agreement between the U. S. Bureau of Mines, the Missouri School of Mines and Metallurgy, Rolla, Mo., and the Eastern Magnesia Talc Co., Burlington, Vt. The Bureau of Mines will welcome reprinting of this report provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Report of Investigations 3314."

2/ Junior metallurgist, Metallurgical Division, U. S. Bureau of Mines.

3/ Research metallurgist, Missouri School of Mines and Metallurgy, Rolla, Mo.

4/ Ladoo, R. B., Talc and Soapstone; Their Mining, Milling, Products, and Uses: Bull. 213, Bureau of Mines, 1923, 133 pp.

5/ MacConkey, C. A., Grading and Uses of Talc; Canadian Chem. and Met., vol. 19, 1935, p. 85.

Talc, a hydrous magnesium silicate, is usually a secondary mineral resulting from alteration of serpentine, tremolite, pyroxenite, magnesite, or dolomite. Alteration may not be complete, and the primary minerals may constitute the gangue. The pure mineral has a specific gravity of 2.7 to 2.8 and a hardness of 1. The mineral rarely occurs in well-developed crystals and usually is found as foliated plates or fibrous aggregates. The compact massive variety of talc is often called steatite, and the impure massive form, which may contain only 50 percent of talc, is referred to as soapstone.

Deposits of talc are widely distributed over the earth's surface and are found in nearly every country. The French, Italian, Japanese, and Indian talcs are noted for their purity. Few domestic talcs can compete with the imported materials in color or purity, and the domestic production is largely used in the bulk trade. The latest available reports of the Bureau of Mines[6/] indicate that the United States produces about 40 percent of the world's talc. New York, Vermont, California, and North Carolina are the principal sources of the domestic supply. The total production in 1934 amounted to about 139,000 tons and represented a value of approximately $1,451,000. The imports, largely from Canada, Italy, and France, were listed at 20,500 tons, with a value of about $426,400.

The talc demanded by industry must have certain physical and chemical characteristics when ground. Usually it is purchased for its physical properties rather than for its purity. The color, shape, size of grains, slip, bonding strength, melting point, vitrification range, and oil-absorption index often are more important to the consumer than purity. Each consumer demands a particular type of material, established as a result of favorable past experience. No uniform or standardized tests for comparison of physical properties exist in the industry. As Ladoo[7/] points out, a series of standard tests for comparing physical properties would go far toward stabilizing the industry and would form a basis for comparing the relative efficiency of various methods of grinding and beneficiation. The physical tests outlined by Ladoo were followed in this investigation. Comparisons for color and slip were made against Canadian and Italian talcs as arbitrarily chosen standards.

The technical developments in the talc industry have been confined largely to improvements in the mechanical preparation of various products. Little or no attempt has been made to beneficiate talc rock by recognized ore-dressing methods. Selective mining when necessary, followed by crushing and grinding dry to 200-mesh or finer by pebble mills or other crushing devices in closed circuit with air separators, has been the customary method of preparation. In certain instances a moderate degree of beneficiation has been obtained by rejecting the separator tailings. Although the talc recovery was low the procedure was justified because it enabled the operator to meet the demands of the trade for specific grades. The disposal of the separator tailings containing 30 to 40 percent of talc became increasingly serious as the demand for the higher grades increased. In certain cases as much as 3 tons of talc rock were required to yield 1 ton of the desired grade. The 2 tons accruing as a low-grade product had relatively little value and was difficult to market.

6/ Emery, A. H., and Stoddard, B. H., Talc and Soapstone: Minerals Yearbook, 1935, U. S. Bureau of Mines, p. 1069.

7/ Ladoo, R. B., work cited, p. 87.

MAGNESITE AS TAILINGS

When the testing was fairly under way it was found that the gangue in the talc ore was magnesite instead of dolomite, as formerly believed. Hence it was anticipated that the tailings might be marketed as a magnesite product and thus constitute a valuable byproduct. The Vermont deposits are favorably located with respect to the magnesite-consuming centers. A certain amount of impurities in the magnesite product is allowable. To produce dead-burned magnesite 8 to 14 percent of impurities - silica, iron, and lime - are essential so that the material will bond under heat. The experiments of Taylor and Rees[8/] indicate that talc forms a better bonding material in magnesite than silica through the formation of forsterite. The reaction is as follows:

$$5\ (MgO \cdot CO_2) + 3\ MgO \cdot 4\ SiO_2 \cdot H_2O = 4\ (2\ MgO \cdot SiO_2) + H_2O + 5\ CO_2.$$
Magnesite Talc Forsterite

The tests using 10 percent of talc as a bond for Greek magnesite produced a good briquet after firing to 1,500° C. The magnesite alone did not make a well-bonded brick. It is possible that tailings containing not more than 10 percent of talc might be suitable for producing dead-burned magnesite, provided the iron and lime contents are satisfactory.

According to Ladoo[9/] and Tyler[10/] the iron content of dead-burned material may be 4.5 to 8 percent of Fe_2O_3. In order to maintain the iron content of the finished product the tailings should contain not more than 3 percent of iron. Attempts to produce tailings of this low iron content were not successful in the present investigation. In the ores tested, a minimum iron content of about 5 percent seemed to be the limit. Float-and-sink tests and roasting and magnetic concentration tests on the tailings indicated that the bulk of the iron was present as ferrous magnesium carbonate and could not be rejected.

The sulphide minerals in the ore account for only a small percentage of the total iron, but they may be recovered by customary sulphide flotation methods. The customary procedure was first to float the talc and then the sulphides. Attempts to reverse the separation were not successful. Talc is more difficult to depress than the sulphides, and any condition that gave good sulphide flotation also gave flotation of the talc. Carnochan and Rogers[11/] report successful flotation of talc from dolomite by using xanthate in conjunction with soda ash and sodium silicate. Had sulphide minerals been present they would have floated with the talc.

The desire of certain operators to recover more of the talc in a product of superior quality resulted in the Eastern Magnesia Talc Co. of Burlington, Vt., entering into a cooperative agreement with the U. S. Bureau of Mines to investigate

---

8/ Tyler, W. H., and Rees, W. J., An Investigation Into the Commercial Utility of a Deposit of Magnesite-Bearing Rock in the Anglo-Egyptian Sudan: Trans. Ceram. Soc. (London), vol. 23, March 1934.

9/ Ladoo, R. B., Nonmetallic Minerals; Occurrence, Preparation, Utilization: McGraw-Hill Book Co., New York, 1925, p. 333.

10/ Tyler, P. M., Magnesite; Bureau of Mines Inf. Circ. 6437, 1931.

11/ Carnochan, R. K., and Rogers, R. A., Experimental Tests On Madoc Talc For The Separation of Dolomite: Canadian Dept. Mines, Mines Branch, Rept. 736, 1934, pp. 231-4.

the concentration of talc ores by ore-dressing methods. The data embodied in this report are the result of that investigation. The laboratory work was done at the Mississippi Valley Experiment Station of the Bureau in cooperation with the Missouri School of Mines and Metallurgy, Rolla, Mo.

LABORATORY WORK

Preliminary Examination

Samples of the crude talc rock and air-separator products of various grades and fineness were obtained from the plant of the cooperating company at Johnson, Vt. Petrographic examination of the samples showed talc associated with a gangue composed principally of a ferrous variety of magnesite. Silicate gangue minerals were not detected in any samples examined. Small amounts of pyrite, pyrrhotite, and a nickel arsenosulphide - gersdorffite - constituted the accessory minerals. Siderite was present in solid solution with magnesite, and only a few discrete grains of clean siderite were detected in any of the samples.

Microscopic examination of sized portions of the crushed rock indicated that fine grinding would be necessary to liberate the talc. This examination also indicated that a beneficiation process adapted to the handling of fine material, such as flotation, was needed. The flotation will be considered under two heads: Small-scale batch flotation and continuous flotation in a small six-cell machine.

Small-Scale Batch Flotation

In most flotation processes the object is to float some nonsiliceous material as the concentrates and to leave the siliceous material in the tailings. With the customary flotation reagents the silicate minerals, as a rule, are the least floatable and may be depressed. Talc is an exception and may be classed as one of the readily floatable minerals. In common with foliated graphite and molybdenite, talc possesses a high inherent floatability. The preliminary work indicated that a frothing agent alone will permit good flotation. Other reagents are necessary, however, to maintain selectivity and to improve the character of the froths. The problem in the flotation is not to float the talc but rather to prevent flotation of the gangue minerals.

The aim in the early work was to recover the talc in concentrates of improved color and slip. Concentrates assaying 95 percent or more of talc were desired, since the color and slip are vitiated by the amount of contaminant present. Although the inherent color of pure Vermont talc is inferior to that of many of the imported talcs, the rejection of the gangue minerals should improve the physical properties and allow the concentrates to be used in many industries in which plant products now used are unsuitable.

The preliminary flotation tests were made in a laboratory machine of the sub-aeration type. A pneumatic cell also was used and gave satisfactory results. Charges of the crude rock were stage-ground in a porcelain jar with flint pebbles to 100- or 200-mesh. Flotation was performed in the grind water, and the pulp density was in accord with usual practice. Double cleaning of the rougher concentrates usually was sufficient to give final talc concentrates of the desired grade.

Direct flotation of the dry-ground plant products was a little more difficult. Gangue particles dusted with talc floated almost as readily as a clean talc particle, and large amounts of depressants or repeated cleanings were necessary to reject the gangue. Although satisfactory results were obtained on several dry-ground products, attention was largely devoted to a study of the flotation of the laboratory wet-ground crude ore.

Little difficulty was experienced in floating talc with any of the customary frothing agents. The choice of frother is largely a matter of individual preference. Pine oil, cresylic acid, and many of the alcohols were used with equal success. Only a small amount of frother is required. A deficiency of frother increases the tailings losses, whereas an excess tends to give low-grade concentrates. With an excess of reagent the froths become voluminous and occlude considerable magnesite. A voluminous pine-oil froth may be modified with kerosene. It gives a compact, heavy froth that is easy to handle. A proper balance of the two reagents is necessary. An excess of kerosene should be avoided, as it causes the collapse of the froth and tends to float the sulphides with the talc.

Kerosene and other relatively insoluble oils serve as collectors for talc. Although the oils may be added directly to the flotation pulp, the use of oil emulsions proved advantageous in the laboratory. The efficacy of the oil apparently was increased, and the amount of oil added could be controlled more easily. The emulsions of oil in water are easily prepared by using a suitable emulsifying agent, such as Emulsol X-1. Concentrated emulsions containing 83 percent of oil, 10 percent of emulsifier, and 7 percent of water are easily prepared and are very stable. Five-percent solutions of these emulsions were used in both the batch and continuous tests.

Talc is so floatable that any of the customary frothing or collecting agents will give flotation. Depression of the gangue must be emphasized. Contamination of the concentrates is largely a result of entrapped gangue particles or locked middling grains. These materials may be depressed in the roughing or cleaning operation by using a suitable reagent. Sodium silicate is effective, and a small amount is sufficient. The choice of depressants depends largely on the collecting or frothing agent employed. Precise control of the silicate is not essential, as it has little tendency to depress the talc unless an excessive amount is used. It is advisable to use a protective alkali with the silicate such as soda ash or caustic soda. Tests with various reagents indicated that talc floats readily with pine oil and kerosene through the pH range of 5.5 to 12.

Depressants other than silicate were investigated, but they had little or no advantage. The organic deflocculating agents were tried, but their merit was doubtful. An effective depressant for both magnesite and sulphides is acid dichromate, but its use is restricted because it discolors the talc. Although the discoloration is dissipated by filtration and washing it is not advisable to use reagents that have a discoloring effect. Cyanides or sulphites were used on several occasions to depress the sulphide minerals, but lowering the iron content of the concentrates was insufficient to justify their use. Had the ores contained an appreciable amount of sulphide minerals the cyanide might have been advantageous.

The results of a typical laboratory test in which soda ash and sodium silicate were used on talc rock stage-ground to pass 200-mesh are given in table 1. Talc concentrates assaying 97 to 98 percent have been produced in other tests, but the recovery was somewhat lower. These concentrates have a much better color and slip than any of the plant products examined but are inferior to some of the imported talcs.

Although the pine oil-kerosene combination is selective with regard to talc, it seemed advisable to attempt the use of customary nonsulphide collecting agents. The fatty acids or their derived soaps were used, and although a fair separation was accomplished the results were generally inferior to those obtained with the nonsoap reagents. The froths were voluminous, and there was a tendency for the magnesite to float with the talc. A rigorous treatment was necessary to produce clean concentrates. The talc content of the tailings was about the same as with the frother-kerosene reagent combination.

TABLE 1. - Flotation of crude talc rock wet-ground to pass 200 mesh

Product	Weight, percent	Assay, percent				Percent of total			
		Talc	Fe	$CaCO_3$	$MgCO_3$	Talc	Fe	$CaCO_3$	$MgCO_3$
Talc concentrates ...	40.3	94.54	0.49	0.02	5.78	81.8	6.2	1.4	4.7
Middlings	6.2	49.24	3.90	.84	46.54	6.5	7.6	8.9	5.8
Rougher concentrates	46.5	88.50	.94	.13	11.21	88.3	13.8	10.3	10.5
Magnesite tailings ..	53.5	10.14	5.12	.98	82.61	11.7	86.2	89.7	89.5
Composite(calculated)	100.0	46.58	3.18	.58	49.41	100.0	100.0	100.0	100.0

Reagents used	Pounds per ton of crude ore		
	Rougher	Cleaner 1	Cleaner 2
Soda ash	1.00	0.5	
Sodium silicate	.20	.2	0.1
Pine oil	.16		
Kerosene	1.00		

A combination of oleic acid and terpineol was used by A. B. Jackson in British Patent 425362 for floating talc from a magnesite gangue. Depression of the magnesite was secured by means of a metal salt - lead nitrate. Such a procedure is worthy of consideration provided the amount of collector is kept at a minimum.

The use of a metal salt and sodium silicate, which had proved effective as a calcite depressant in scap flotation,12/ was also effective for magnesite. Other depressants were used, but in no case were the results superior to those obtained with the frother-kerosene combination.

A reagent that has proved beneficial in nonsulphide flotation is Calgon - sodium hexametaphosphate. This material forms soluble complex salts with hard-water constituents and has been used as a water softener. Calgon forms soluble

12/ Clemmer, J.B., and O'Meara, R.G., Flotation and Depression of Nonsulphides; Calcite, Silicate and Silicates, Fluorspar, Barite, Apatite, and Tungsten Minerals: Rept. of Investigations 3239, Bureau of Mines, June 1934, pp. 9-26.

complex salts with the calcium and magnesium soaps and probably serves as a depressant for calcium and magnesium minerals by preventing the formation of insoluble soap coatings on the mineral particles. In addition to being a depressant for magnesite it was also of value as a deflocculating agent and was used with the pine oil - kerosene reagent combination. Calgon did not retard the talc materially when used in amounts of less than 2 pounds per ton.

Another group of compounds that possesses merit in talc flotation is the sodium or ammonium alkyl sulphates. Emulsol X-1 belongs to this group and has been discussed in a previous paper.[13/] Emulsol X-1 is a powerful frother and may be used instead of pine oil. Although its froths are voluminous, they may be modified with kerosene. The use of X-1 as an emulsifying agent has been mentioned already.

Continuous Flotation in a Small 6-Cell Machine

The batch flotation tests made in the laboratory machine indicated that satisfactory concentration of the talc could be effected by several reagent combinations. The preferred reagent charge was a frother in combination with kerosene and an alkaline dispersing reagent. The next step in the investigation was to attempt the separation under conditions simulating plant practice. Ton lots of air-separated rejects and of crude rock were obtained. The tests were made in a small pilot plant consisting of a grinding and a flotation unit. The feed rate was maintained at 60 or 120 pounds per hour, depending on the circuit employed. The ball mill was in closed circuit with a reciprocating drag classifier of the Esperanza type. The grinding capacity was considerably higher than that of the flotation unit and allowed considerable leeway in preparing the ores for flotation.

Both samples had been ground to a particle size suitable to feed directly to the 19- by 36-inch pebble mill. Sillimanite balls were used as the grinding medium and performed satisfactorily. Although the mill was not lined, no discoloration of the talc was noted after the mill had been run long enough to remove accumulated rust.

The sizing analyses of the flotation feeds for typical tests on the two ores, together with the original ball-mill feed, are given in table 2. The flotation feeds were of about the same fineness. Comparative grinding tests indicate that the feeds had a grinding rate five times that of chert and twice that of dolomite.

The flotation unit consisted of a six-cell subaeration machine, together with a pump, conditioner, and accessory reagent feeders. The flotation feed (classifier overflow) was pumped to the mechanical conditioner, thence it flowed by gravity to the cells. A 5-minute conditioning was ample. The six-cell unit was flexible and permitted several circuits to be tried. The customary arrangement was to use four cells as roughers and the remaining two cells for double cleaning. The middlings were returned to the roughing cells. In certain tests scavenger concentrates were produced on the last roughing cell and withdrawn from the circuit. Enough cells were not available to re-treat the scavenger concentrates.

TABLE 2. - Sizing analyses of typical products on two types of talc ore

Size, mesh	Air separator rejects		Crude talc rock	
	Original feed	Flotation feed	Original feed	Flotation feed
8	0.4	.....	0.1	.....
10	2.7	.....	1.4	.....
14	10.4	.....	15.0	.....
20	20.3	.....	12.5	.....
28	20.1	.....	8.9	.....
35	16.6	.....	8.9	.....
48	13.2	0.2	9.0	0.3
65	8.9	1.6	6.7	2.3
100	5.1	10.5	8.9	10.1
150	1.4	17.2	6.3	14.7
200	.2	14.4	5.1	12.4
-200	.7	56.1	17.2	60.2
Total	100.0	100.0	100.0	100.0

The flotation feed was held at 25 percent solids. With four cells as roughers, a feed rate of 120 pounds per hour gave a roughing period of about 10 minutes. This was sufficient for satisfactory flotation. The feed rate was reduced to give an equivalent roughing period on those tests in which a scavenger product was made. Although a higher feed rate might have been used, the crowding of the machine is not good practice: Obtaining additional capacity by increasing the density of the pulp should not be attempted, as occlusion in the froth is increased and the selectivity is impaired.

One of the objections often raised against small-scale continuous testing is the lack of precise control of reagents. The inability to add the reagents in small amounts may be overcome by using feeders of the proper design. Satisfactory control of reagents was obtained by using feeders of the floating constant-head siphon type. The siphons were made from 1/4-inch laboratory glass tubing and were constricted to form a jet at the discharge end. The required size of the jet was obtained by trial; a maximum flow of 30 cc per minute and a minimum flow of somewhat less than 1 cc was ample for the reagents used. The delivery of a given jet was regulated by adjusting the siphon in the float to change the head. Once the head for a given flow was established, the feeder required little attention and delivered a uniform flow of reagent for long periods.

All of the reagents, kerosene and pine oil excepted, were water-soluble and were added to the circuit as 1-percent or 5-percent solutions. Kerosene was added in either the natural or an emulsified condition, and pine oil was added as a dilute emulsion. Pine-oil emulsions are not as stable as kerosene emulsions and cream on standing. This creaming was minimized by preparing small amounts of the dilute emulsion frequently.

The type of froth and the appearance of the products served as a guide in regulating the quantity of reagent necessary and the place of addition. Frequent use was made of the vanning plaque. Each test lasted long enough to justify the conclusion that a balanced condition had been attained in the circuit before sampling. Time samples of the products were taken to ascertain the weight distribution. Comparison of the calculated and actual feed analyses was an additional check on the tests.

The pilot-plant tests on the air-separator rejects and the crude rock gave nearly identical recoveries. Conditions favorable for one were favorable for the other also. The concentrates from the crude ore had slightly better color and slip than those from the separator product.

The tests indicated that the combination of a frother, kerosene, and an alkaline dispersant would give satisfactory recovery. Roughing, followed by double cleaning and returning the middlings to the roughing circuit, gave a 90-percent recovery in concentrates assaying 92 percent talc and 1 percent iron. The tailings assayed 12 percent talc. Microscopic examination of the concentrates showed that the contamination was largely middling grains coarser than 200-mesh and indicated a need for finer grinding or more rigorous cleaning. The fine portion of the concentrates possessed a marked superiority in both color and slip. The tailings losses were largely in the minus 400-mesh size.

In subsequent tests the frother was increased to recover more of the fine talc and produce tailings containing less than 10 percent of talc. Although low-talc tailings were made, the grade of the concentrates was reduced by occlusion of fine magnesite. Increasing the dispersant in the cleaners to give the desired grade of concentrates resulted in greater tailing losses. The tests indicated that the locked middlings must be removed from the circuit if high-grade concentrates and low-grade tailings were to be produced easily and consistently. It was felt that such a procedure might enable a large proportion of the iron to be rejected and allow talc concentrates and magnesite tailings of lower iron content to be produced. Two circuits were optional: Rejection of the cleaner middlings or rejection of the scavenger concentrates. Tests were made to determine the merits of each procedure.

A typical test in which the middlings were rejected is given in table 3. The concentrates are of satisfactory grade and have good color and slip. An additional cleaning should increase the grade without materially sacrificing recovery. The magnesite tailings are satisfactory as regards talc content but contain too much iron. The rejected middlings contain both fine talc and coarse middling grains. In a large plant re-treatment of this product would be advisable. An appreciable quantity of the talc would be recovered by regrinding and floating. In the present work considerable spray water was required to break down the rougher froth, hence, when the middlings were rejected a higher pulp density was maintained in the roughing cells.

The results of a test in which the scavenger concentrates were rejected are given in table 4. The screen analyses of the products are given in table 5, and the distribution of talc and iron in the various products is recorded in table 6 according to size. Double cleaning was practiced, and the middlings were returned to the roughing circuit. The scavenger concentrates were produced in the last two

cells, xanthate and additional frother being used to float the sulphide minerals, pyrite, pyrrhotite and gersdorffite, and the remaining talc. Although the sulphide minerals floated readily, the iron content of the magnesite tailings was not reduced, as most of the iron occurs as ferrous magnesite. Although the talc content of the tailings is reduced to a satisfactory point, the screen analyses show the losses to occur largely in the fine sizes. The loss of fine talc may be reduced by increasing the quantity of frother or by prolonging the flotation period. If the magnesite recovery is less important than the grade, mechanical means may be used to reject the fine talc from the tailings; a rough sedimentation test in the laboratory, equivalent to desliming the tailings in a bowl classifier, gave final magnesite tailings assaying 4.4 percent of talc and 7.63 percent of iron.

TABLE 3. - Pilot-plant test on crude talc rock in which cleaner middlings were rejected

Product	Weight, percent	Assay, percent			Percent of the total		
		Talc	Fe	$MgCO_3$	Talc	Fe	$MgCO_3$
Feed		53.85	2.94	42.49			
Talc concentrates.	41.6	93.89	.56	5.21	74.1	7.5	5.0
Middlings	23.3	51.38	2.16	45.06	22.7	16.1	24.2
Magnesite tailings	35.1	4.68	6.78	87.59	3.2	76.4	70.8
	100.0	52.67	3.12	43.41	100.0	100.0	100.0

Reagents used	Pounds per ton of crude ore		
	Rougher	Scavenger	Cleaner
Soda ash	1.8	.....	.....
Sodium silicate	.5	.....	0.2
Pine oil	.16	0.08	.....
Kerosene	2.5	.....	.....

TABLE 4. - Pilot-plant test on crude talc rock in which the scavenger concentrates were rejected

	Weight, percent	Assay, percent				Percent of total			
		Talc	Fe	$CaCO_3$	$MgCO_3$	Talc	Fe	$CaCO_3$	$MgCO_3$
Feed		53.61	3.57	2.60	40.63				
Talc concentrates.	48.8	94.82	.29	.30	4.61	86.6	4.0	5.6	5.8
Scavenger concentrates	10.2	37.85	4.11	1.35	36.53	7.2	11.8	5.2	9.6
Magnesite tailings	41.0	7.97	7.26	5.71	80.21	6.2	84.2	89.2	84.6
Composite (calc.).	100.0	53.40	3.54	2.63	38.86	100.0	100.0	100.0	100.0

Reagents used	Pounds per ton of crude ore		
	Rougher	Scavenger	Cleaner
Sodium silicate	0.4	.....	0.2
Pine oil	.14	0.08	.....
Kerosene	2.2	.3	.....

TABLE 5. - Screen analyses of flotation products

Product	Size, mesh	Weight, percent	Assay, percent		Percent of total	
			Talc	Fe	Talc	Fe
Cleaner concentrates	Plus 200	39.7	91.8	0.33	38.6	44.5
	Plus 400	25.5	94.8	.31	25.5	26.9
	Minus 400	34.8	97.6	.24	35.9	28.6
	Composite (calc)	100.0	94.6	.29	100.0	100.0
Scavenger concentrates	Plus 200	47.3	40.4	4.46	49.9	57.1
	Plus 400	15.2	48.2	5.40	19.1	22.2
	Minus 400	37.5	31.6	2.04	31.0	20.7
	Composite (calc)	100.0	38.3	3.70	100.0	100.0
Tailings	Plus 200	35.2	3.4	7.13	14.6	35.3
	Plus 400	28.6	3.4	7.56	12.0	30.4
	Minus 400	36.2	16.4	6.75	73.4	34.3
	Composite (calc)	100.0	8.11	7.12	100.0	100.0
Feed	Plus 200	38.7	52.3	3.36	37.9	38.1
	Plus 400	25.7	50.3	3.93	24.2	29.3
	Minus 400	35.6	56.8	3.15	37.9	32.6
	Composite (calc)	100.0	53.4	3.44	100.0	100.0

TABLE 6. - Distribution (percentage) of talc and iron according to size

	Plus 200 mesh		-200 +400 mesh		Minus 400 mesh	
	Talc	Iron	Talc	Iron	Talc	Iron
Talc concentrates	87.9	4.9	91.1	3.9	82.0	3.6
Scavenger concentrates ..	9.6	16.4	5.8	8.3	6.0	7.0
Magnesite tailings	2.5	78.7	3.1	87.8	12.0	89.4
Feed	100.0	100.0	100.0	100.0	100.0	100.0

The contamination in the concentrates is due largely to coarse middling particles. More rigorous cleaning will yield concentrates assaying from 97 to 98 percent of talc. The middling particles segregate in the middlings or scavenger product and can be isolated for re-treatment.

The concentrates are considerably coarser than the talc customarily marketed. Dry grinding to pass 200-mesh or finer improved both the color and slip and gave a product that compared favorably with several of the imported talcs. The talc and iron assays and the percentage of minus 400-mesh material in samples of imported talc are given in table 7.

TABLE 7. - Analyses of several imported talcs

Kind	Minus 400-mesh, percent	Assay, percent	
		Talc	Iron
Japanese talc ..	99.5	96.42	0.05
Canadian talc...	67.5	87.79	.26
Indian talc	75.2	98.47	.07

The scavenger concentrates were composed largely of talc middling particles and sulphides. They contain about 0.5 percent of nickel, which can be recovered by re-treatment. Due to their high gravity, the sulphide minerals tend to segregate in the classifier sand. Tabling of this product was practiced in several tests, but the reduction in iron content was insufficient to justify the operation. Had an appreciable amount of iron sulphides or oxides been present, tabling of the drag sand and flotation of the sulphides in the scavenger treatment would have been helpful. Although only a small quantity of gerdsorffite is present, the tabling of the drag sand gave a small bulk of concentrates which on recleaning assayed 27.84 percent nickel and 1.63 percent cobalt.

CONCLUSIONS

Small-scale batch tests and pilot-plant flotation tests on Vermont talc-magnesite ores show that these ores may be beneficiated by flotation to make talc concentrates and magnesite tailings. Talc floats readily with a variety of reagents and may be recovered in a high-grade product of improved slip and color. The magnesite tailings may be suitable for certain uses and thus constitute an additional source of revenue to the talc producer.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

2.09
32mo
op2

R. I. 3315 November 1936

DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES
JOHN W. FINCH, DIRECTOR

REPORT OF INVESTIGATIONS

WASHABILITY STUDIES OF COAL FROM THE HENRY ELLEN BED AT ACMAR NO. 5 MINE, ACMAR, ALA.

BY

B. W. GANDRUD AND G. D. COE

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES, USING THE OFFICIAL MAILING LABEL ON THE INSIDE OF THE BACK COVER.

R.I. 3315
November 1936

REPORT OF INVESTIGATIONS

DEPARTMENT OF THE INTERIOR - BUREAU OF MINES

WASHABILITY STUDIES OF COAL FROM THE HENRY ELLEN BED AT ACMAR NO. 5 MINE, ACMAR, ALA. [1]

By B.W. Gandrud[2] and G.D. Coe[3]

INTRODUCTION

The Southern Experiment Station of the U.S. Bureau of Mines, cooperating with the University of Alabama and Alabama coal producers, has made a series of investigations of the washability of coal from commercially important beds of Alabama. The results of 13 investigations pertaining to the Mary Lee, Clark, Black Creek, Jefferson, Pratt, Corona, Carter, Brookwood, Blue Creek, Thompson, and Woodstock beds have been published.[4] This report, the fourteenth of the series, deals with run-of-mine coal from the Henry Ellen bed at the Acmar No. 5 mine of the Alabama Fuel & Iron Co.

The Acmar mine is 1/2 mile west of Acmar, Saint Clair County, in the northern part of the Cahaba field. Present production at the mine is about 800 tons per day, all of which, except the lump coal, is treated in jigs. The washability data were obtained by means of screen-sizing and float-and-sink tests. There are included, also, tests showing the extent of liberation of impurities by recrushing the coarser sizes of coal from the top bench and tests showing the flakiness in selected sizes.

SAMPLING AT THE MINE

The Henry Ellen bed at Acmar consists of two benches separated by a shale parting. At the point of sampling the top bench was 6 feet 3 inches thick and contained a 1-inch and a 1 1/2-inch band of bone. The bottom bench was 5 feet thick and was separated from the top bench by 20 inches of shale. In taking the samples for the washability study the working face was squared up, and a representative 2-ton section was cut from each bench. The shale parting was discarded, according to usual mining practice.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Report of Investigations 3315."

2/ Supervising engineer, U.S. Bureau of Mines, Southern Experiment Station, Tuscaloosa, Ala.

3/ Junior metallurgist, U.S. Bureau of Mines, Southern Experiment Station, Tuscaloosa, Ala.

4/ U.S. Bureau of Mines Reports of Investigations 3012, 3014, 3067, 3083, 3101, 3157, 3165, 3170, 3200, 3204, 3206, 3209, and 3234.

ANALYSIS

In table 1 are recorded the proximate and ultimate analyses of face samples of the two benches taken at the same place in the mine as the washability samples. These data show that the top bench contains more than twice as high a percentage of ash and a little more sulphur than are contained in the bottom bench. The percentages of the other indicated constituents, as calculated to a moisture- and ash-free basis, are essentially the same for the two benches.

Table 2 shows the sulphur analyses and ash-softening temperatures for a complete series of float-and-sink fractions of the 3/4-inch by 3/16-inch raw coal from the combined top and bottom benches. These float-and-sink fractions were made up in such a way as to represent the actual products that would be obtained from a sample in which the top-bench and bottom-bench coal are mixed in weights proportional to the thickness of the two benches. Although no definite relationship exists between specific gravity and either sulphur content or softening temperature of the ash, the coal of lower specific gravity than 1.50 contains decidedly less sulphur than fractions of higher specific gravities, and the ash-softening temperature tends to decrease as the sulphur increases.

SCREEN-SIZING AND FLOAT-AND-SINK TESTING

The screen-sizing and float-and-sink data of tables 3,4, and 5 are illustrated graphically in figures 1 to 19, inclusive, The methods of arranging these data and of plotting the graphical representations are as described in the first of this series of reports.5/ Table 3 shows that, with regard to screen size, the top-bench coal has a fairly uniform ash content - the minimum of 16.8 percent is in the 3/4- to 3/16-inch size and the maximum of 19.9 is in the 35- to 100-mesh size. The bottom-bench coal, however, varies greatly in ash content for the different sizes. Table 4 shows that this range is from 6.9 percent ash in the 4- to 1 1/2-inch size to 16.3 percent in the minus 100-mesh size. A detailed discussion of tables 3 and 4 will not be attempted, as the significance of the data is shown quite clearly by the washability curves.

WASHABILITY CURVES

Figures 1 and 2 are the screen-sizing curves of the top and the bottom benches, respectively. Figures 3 to 10, inclusive, are the washability curves of the various size fractions of the top-bench raw coal. Figures 11 to 18, inclusive, are the corresponding washability curves for the bottom-bench raw coal, and figure 19 shows the washability curves for the total bed. In these figures the cumulative curve shows the weight percent of recoverable float coal for any given ash content. The elementary curve shows the ash content of the highest-ash material included in the float coal, and the specific-gravity curve shows the specific gravity at which the separation between float

5/ Bird, B.M., and others, Washability Studies of the Mary Lee Seam at Lewisburg, Ala.: Rept. of Investigations 3012, Bureau of Mines, 1930, pp. 3-7.

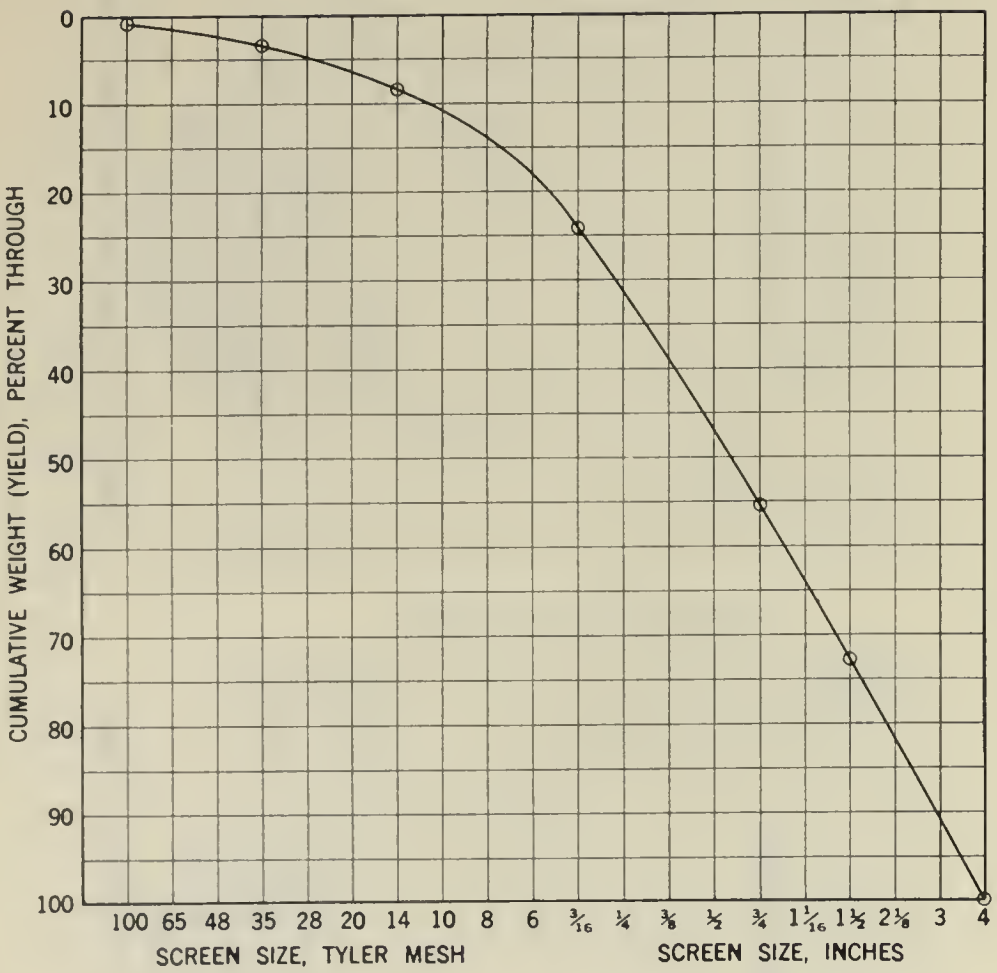


Figure 1.—Screen-sizing test of raw coal from top bench of Henry Ellen bed, Acmar mine.

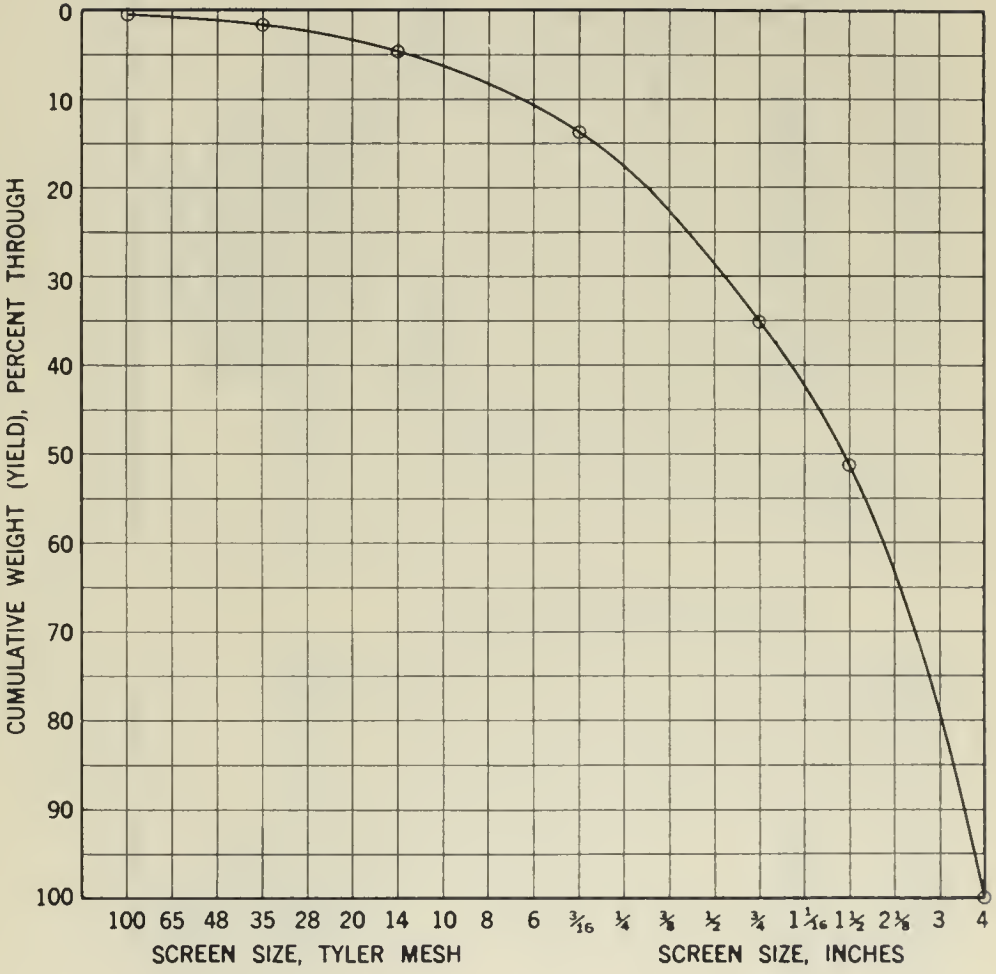


Figure 2.—Screen-sizing test of raw coal from bottom bench of Henry Ellen bed, Acmar mine.

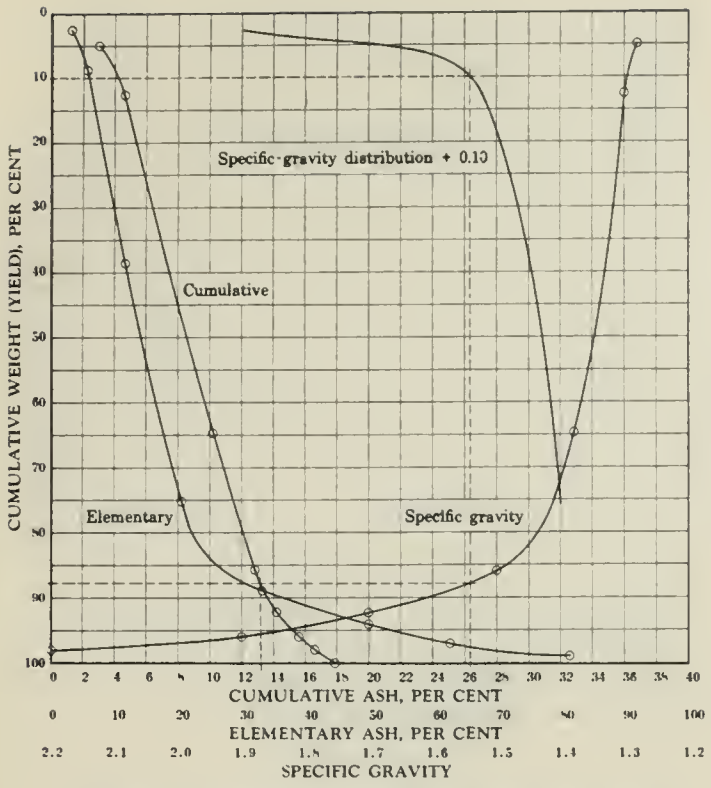


Figure 3.—Top-bench raw coal, composite through 4-inch mesh.

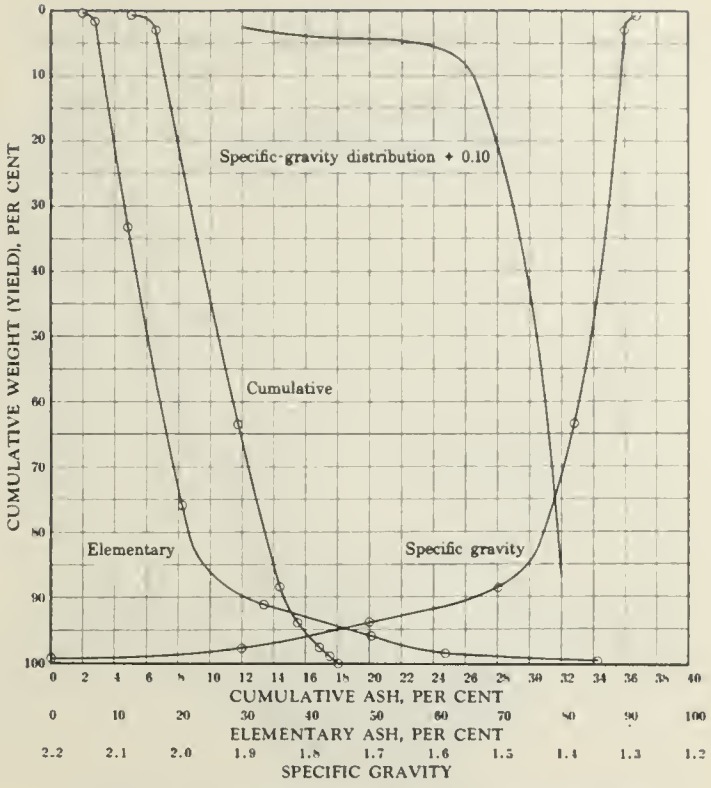


Figure 4.—Top-bench raw coal, through 4-inch on 1½-inch.

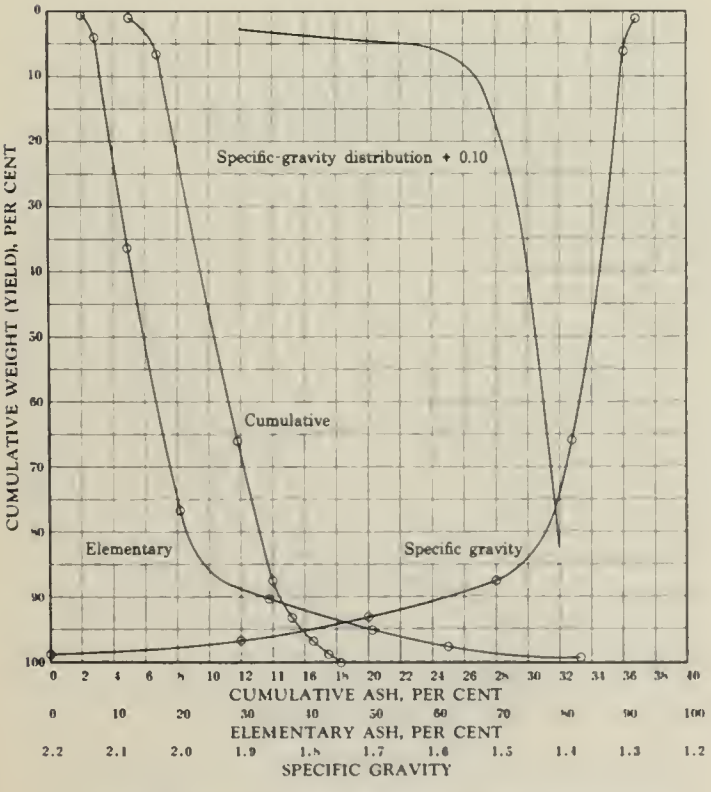


Figure 5.—Top-bench raw coal, through 1½-inch on ¾-inch.

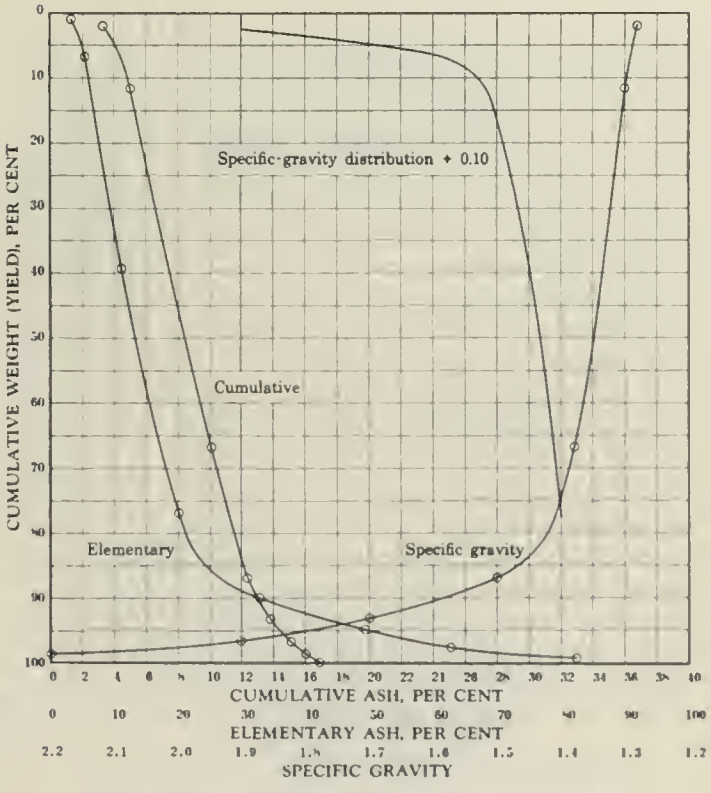


Figure 6.—Top-bench raw coal, through ¾-inch on $\frac{3}{16}$-inch.

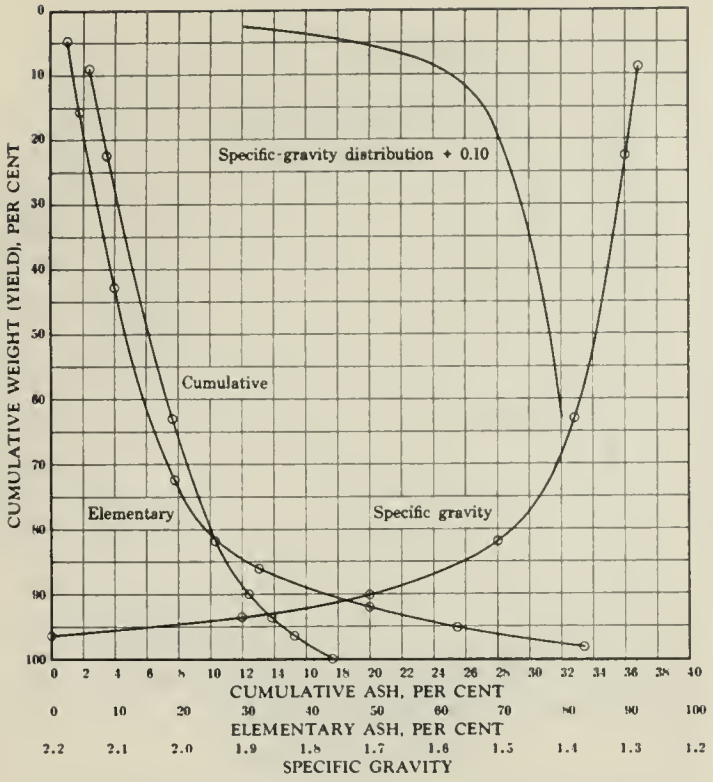


Figure 7.—Top-bench raw coal, through $\frac{3}{16}$inch on 14-mesh.

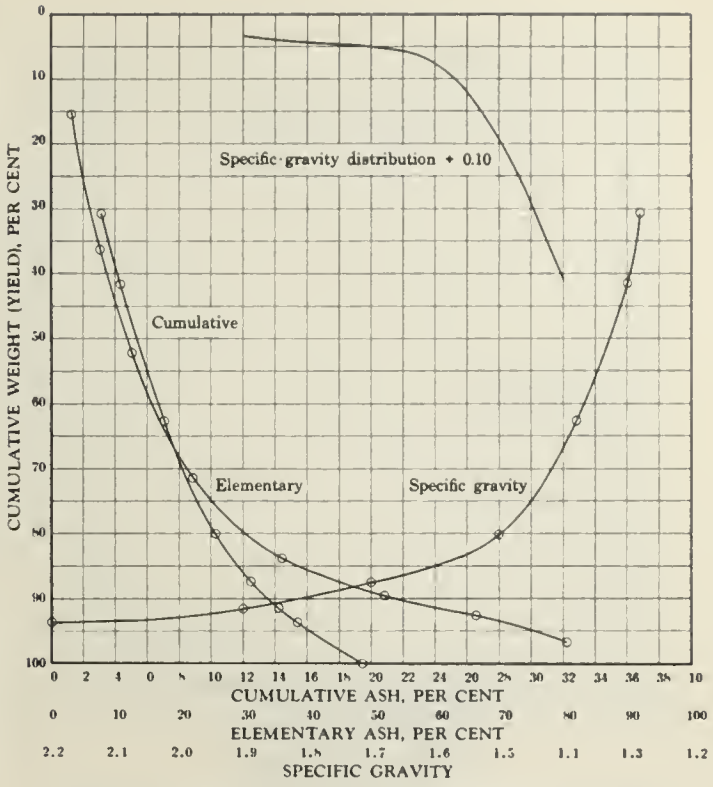


Figure 8.—Top-bench raw coal, through 14-mesh on 35-mesh.

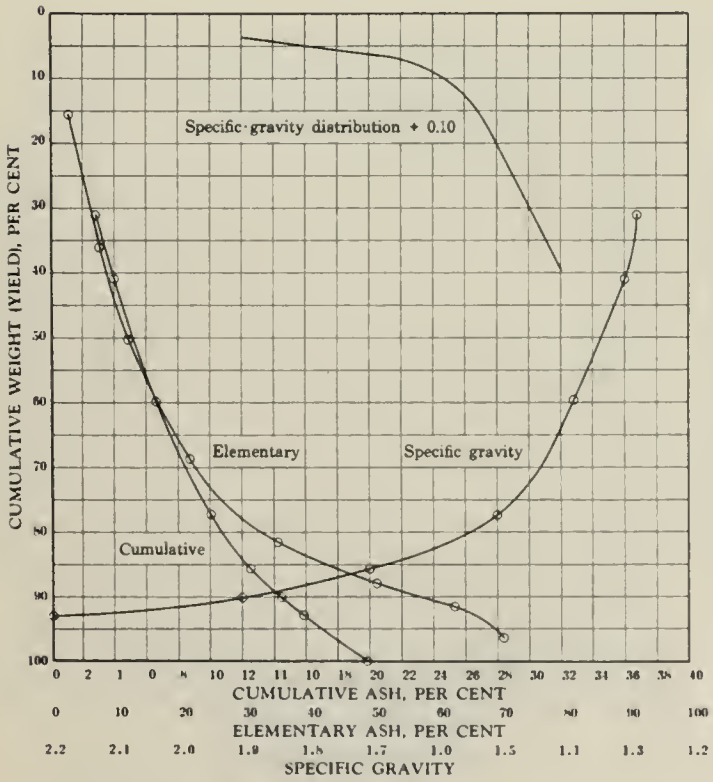


Figure 9.—Top-bench raw coal, through 35-mesh on 100-mesh.

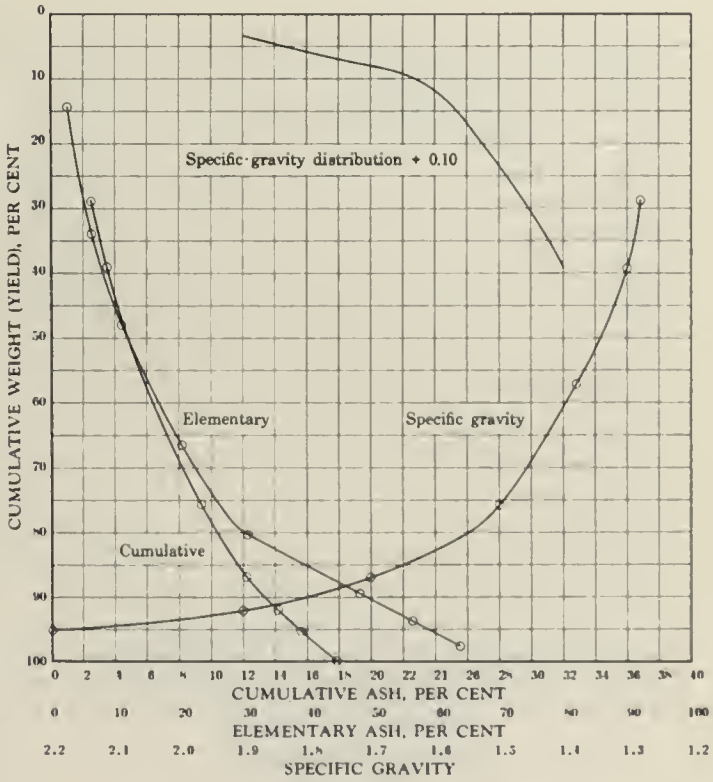


Figure 10.—Top-bench raw coal, through 100-mesh.

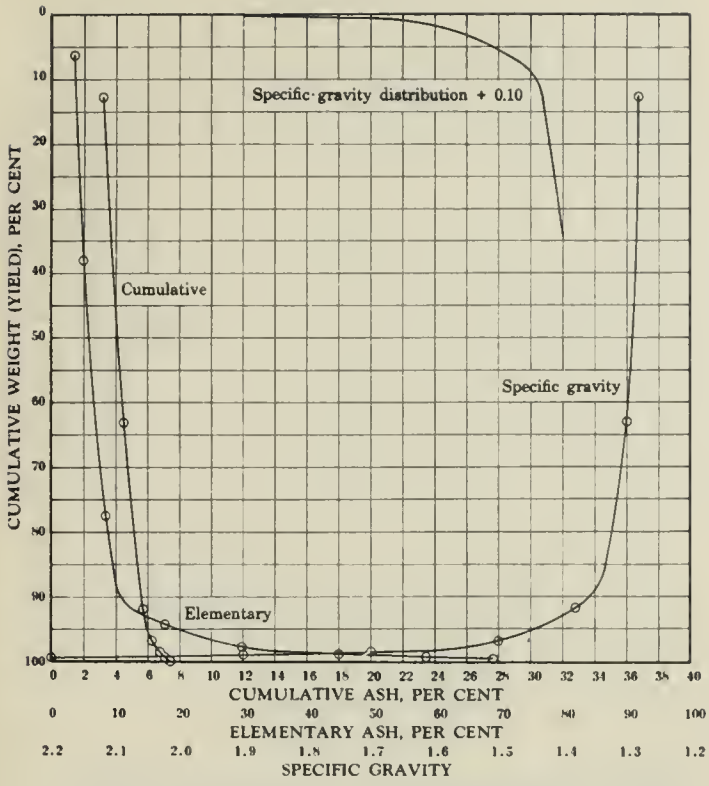


Figure 11.—Bottom-bench raw coal, composite through 4-inch mesh.

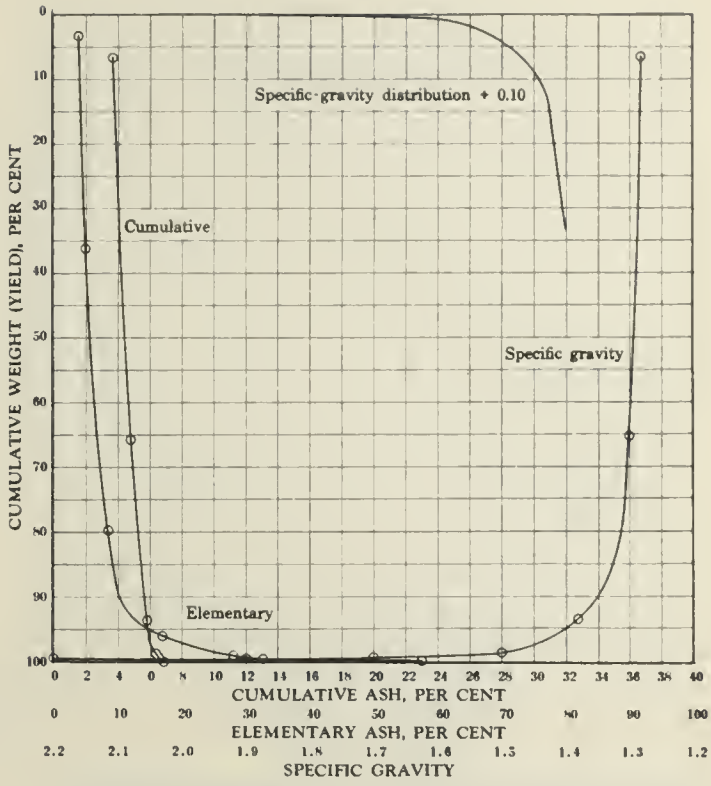


Figure 12.—Bottom-bench raw coal, through 4-inch on 1½-inch.

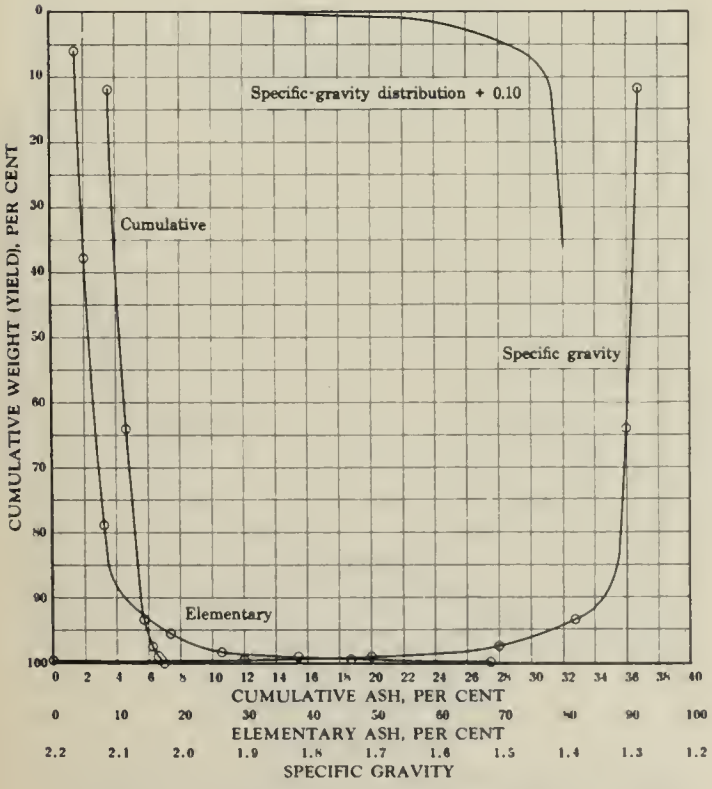


Figure 13.—Bottom-bench raw coal, through 1½-inch on ¾-inch.

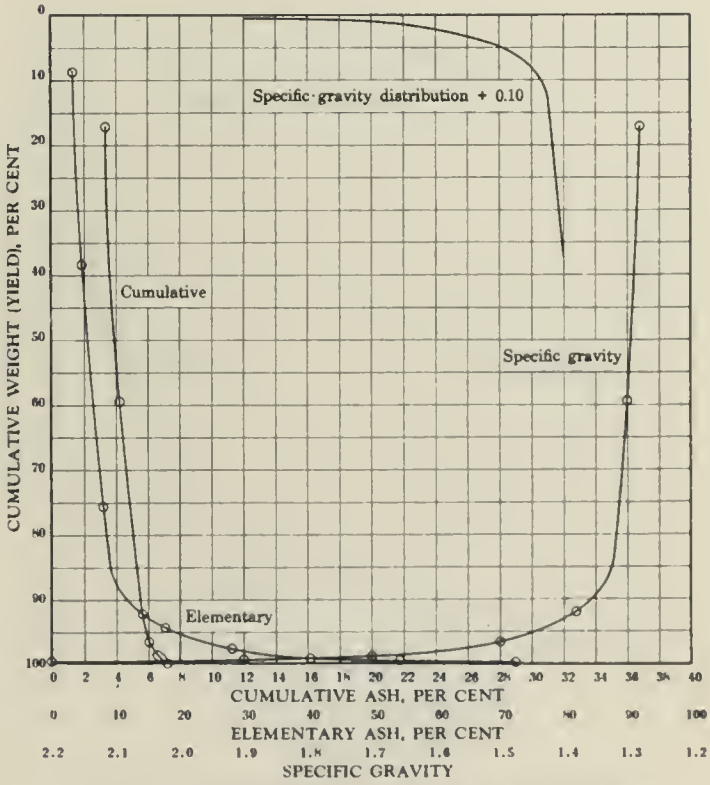


Figure 14.—Bottom-bench raw coal, through ¾-inch on ⅜-inch.

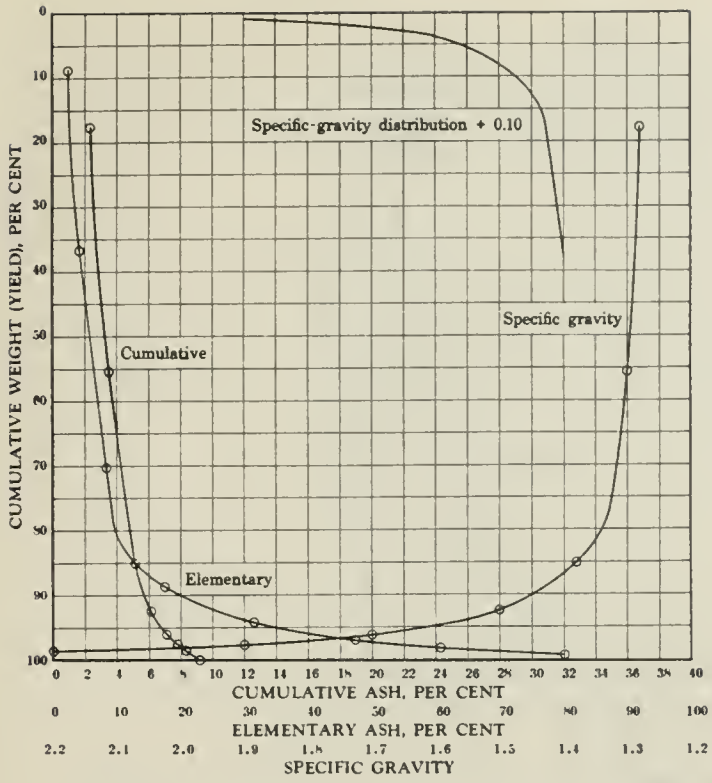


Figure 15.—Bottom-bench raw coal, through $\frac{3}{16}$-inch on 14-mesh.

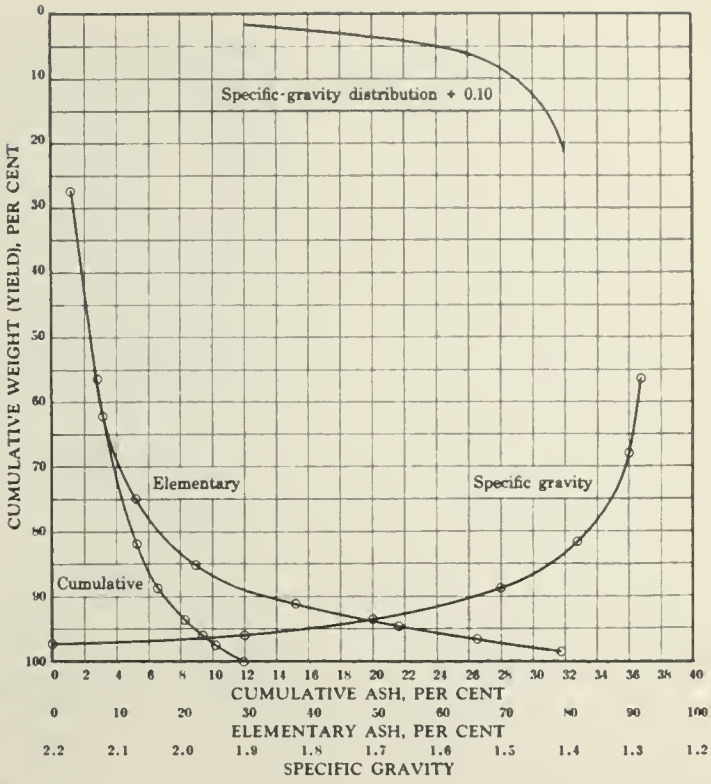


Figure 16.—Bottom-bench raw coal, through 14-mesh on 35-mesh.

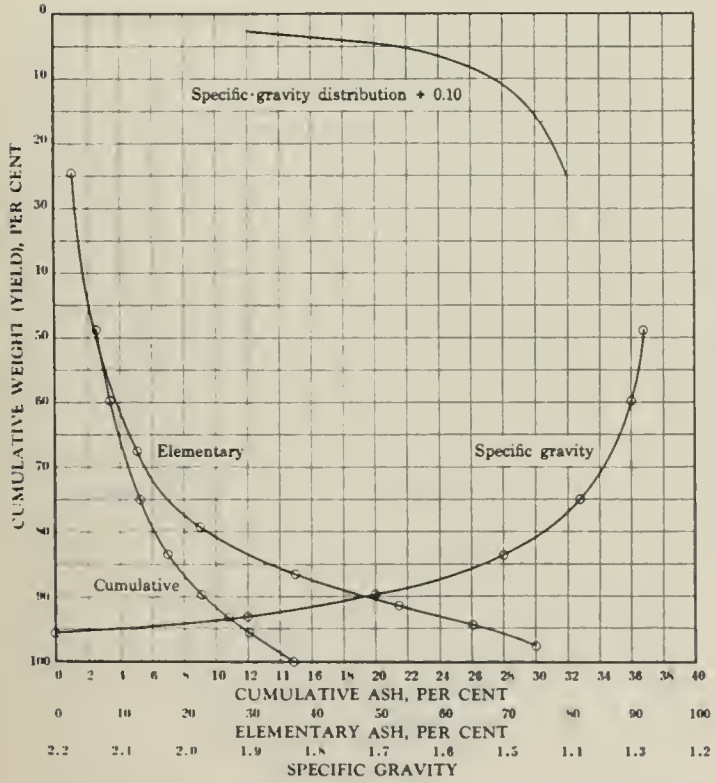


Figure 17.—Bottom-bench raw coal, through 35-mesh on 100-mesh.

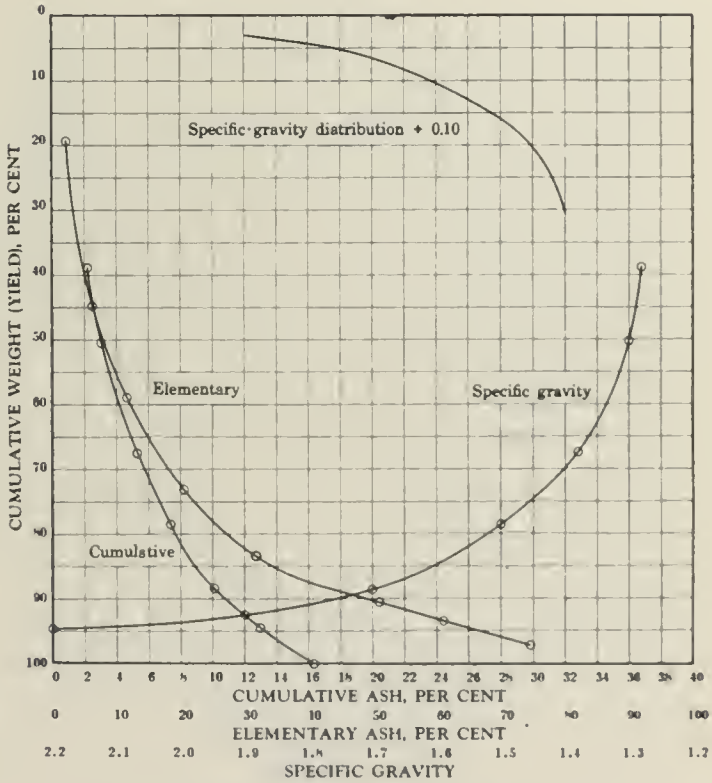


Figure 18.—Bottom-bench raw coal, through 100-mesh.

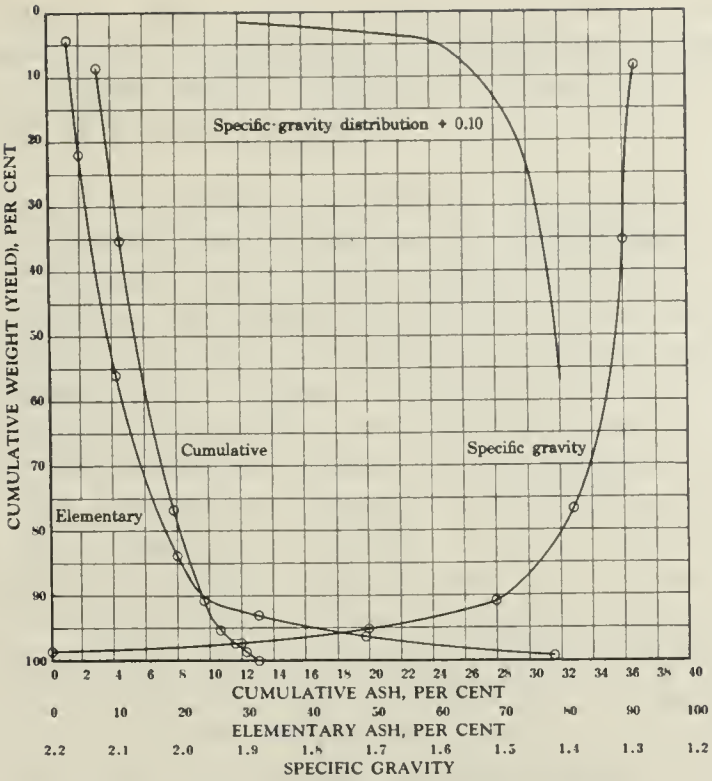


Figure 19.—Raw coal, total bed, composite through 4-inch.

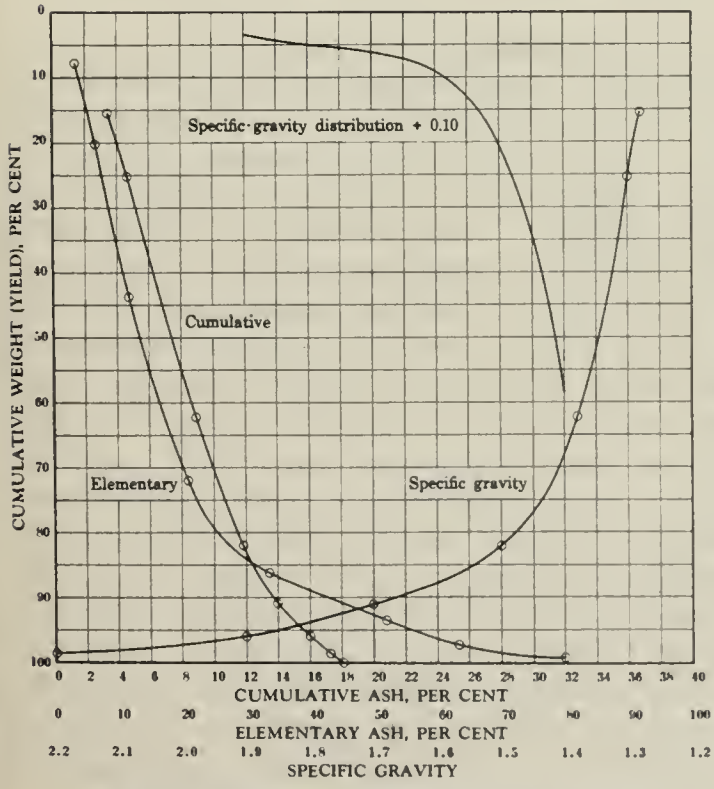


Figure 20.—Raw coal through 4-inch on 1½-inch crushed to pass 3/16-inch, from top bench of mine.

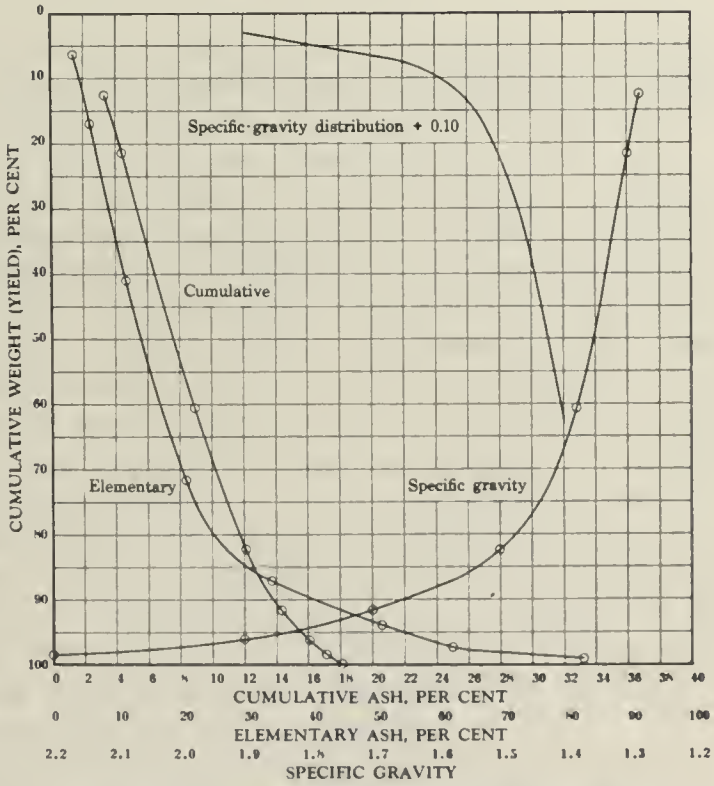


Figure 21.—Raw coal through 1½-inch on ¾-inch crushed to pass 3/16-inch, from top bench of mine.

coal and refuse is made. The ± 0.10 specific-gravity distribution curve shows the relative difficulty of making such a separation. The interpretation of the specific-gravity distribution curves is explained in the following excerpt from a recent Bureau of Mines report:6/

> Previous work has demonstrated the advantage of using the ± 0.10 specific-gravity distribution curves in assigning a value to the relative difficulty of a separation between washed coal and refuse at any particular specific gravity. Although these values are based entirely upon float-and-sink data and do not take into consideration other factors, such as size relationship, the shape of the particles, type of cleaning equipment, and so forth, they give a fair indication of the difficulty of the washing problem and form the basis of the present interpretation. The difficulty of any separation depends primarily upon the amount of material present which has a specific gravity close to the specific gravity at which the separation between washed coal and refuse is to be made. In other words, if a separation is to be made at 1.50 specific gravity, it is the material between 1.40 and 1.60 specific gravities which will cause most of the overlapping of coal and refuse. In general, a separation at a specific gravity corresponding to a reading of ten percent or less on the ± 0.10 curve is one which can be easily made with a very high efficiency. A separation at a specific gravity equivalent to 15.0 percent on the ± 0.10 curve is ordinarily one of moderate difficulty and the recovery and efficiency of the operations may be proportionately lower than those possible in the first case. A separation at a specific gravity corresponding to a reading of 20.0 percent on the ± 0.10 curve is ordinarily one of decided difficulty which can only be accomplished by means of unusually elaborate and very efficiently operated washing systems.

The method of reading the curves is illustrated in figure 3, which represents the top bench coal. The broken line shows that 10 percent ± 0.10 corresponds to a specific gravity of 1.543. Following across from the specific gravity curve at this point, it is shown that the theoretical yield is 87.8 percent, and that this yield, read on the cumulative-ash curve, corresponds to 13.2 percent ash. Applying this same procedure to figure 11, which represents the bottom-bench coal, we find that a separation at the specific gravity corresponding to 10 percent on the ± 0.10 curve gives a yield of 94.7 percent at an ash content of 5.9 percent. Consequently, an average jig plant under normal conditions should wash the top-bench coal to 13.2 percent ash and the bottom-bench coal to 5.9 percent with very little loss of good coal in the refuse. The yield of washed coal would be slightly less, of course, than the theoretical yields of 87.8 and 94.7 percent shown on the cumulative curves for the top bench and bottom bench, respectively. The actual washery yields, however, should not be less than 95 percent of the theoretical yields at the washed-coal ash contents of 13.2 and 5.9 percent. Table 6 summarizes washability data for each bench and for the composite of both benches. The values in this table were read from the washability curves at points corresponding to 10 percent and 15 percent on the ± 0.10 specific-gravity-distribution curves.

6/ Gandrud, B.W., and others, Washability Studies of the Blue Creek Bed at the Connellsville Mine, Connellsville, Ala.: Rept. of Investigations 3200, Bureau of Mines, 1933, 10 pp.

RECRUSHING TESTS

Samples of the 4- to 1-1/2-inch and the 1 1/2- to 3/4-inch top-bench raw coal were crushed to pass a 3/16-inch screen to determine to what extent the ash-forming impurities would be liberated from the coal by such crushing. From float-and-sink tests of the resulting products were obtained the data of table 7. Figures 20 and 21, which were plotted from these data, show that 10 percent on the ± 0.10 curve corresponds to 12.9 percent ash at a theoretical yield of 87.3 percent from the 4-inch by 1-1/2-inch recrushed coal. Corresponding values for the 1 1/2- by 3/4-inch recrushed coal are: Ash, 13.2 percent; and yield, 87.8 percent. Assuming that the difficulty of separation between washed coal and refuse is the same for both crushed and uncrushed coal at specific-gravity points that give the same readings on the ± 0.10 curves, the crushing of the 4- to 1-1/2-inch size would give an additional ash reduction in the washed coal from this size of 1.7 percent. For the 1 1/2- to 3/4-inch size this reduction would be 1.0 percent. The total reduction for the composite top-bench coal by crushing to 3/16-inch instead of 4-inch size would be only 0.7 percent.

FLAKINESS TESTS

The flakiness test is a method to determine the relative sizes and shapes of the particles that comprise the specific-gravity fractions obtained as float-and-sink products. The procedure consists of first screening the float-and-sink fractions on a series of standard sieves with square-mesh openings varying as the $\sqrt{2}$. After each square-mesh product has been weighed it is screened on three sieves with rectangular openings of widths approximately 100, 75, and 50 percent, respectively, of the openings of the square-mesh screen upon which the product was retained. In selecting these sieves the aim is to get meshes with lengths at least as great as the diagonal of the opening of the square mesh through which the product has been screened previously.

The relative flakiness of a product is indicated by the proportion that passes each of the three rectangular-mesh sieves. The material which passes the 50.0 percent openings is decidedly flaky and corresponds approximately to what is known generally as "rash." It predominates in particles of intermediate specific gravity, which, if present in large quantities, complicates the washing problem. As the flakiness of a material as friable as coal cannot be determined with accuracy, the results only have value in so far as general tendencies are exhibited.

The flakiness test usually is made on the 3/4- by 3/16-inch and on the 3/16-inch by 14-mesh sizes of material of 1.38 to 1.90 specific gravity. The inclusion in the washed coal of flaky particles of lower specific gravity than 1.38 ordinarily does no harm, and flakes heavier than 1.90 specific gravity are easily rejected by the washer. The results of the flakiness tests are shown in tables 8 and 9. They show that the percentage of flaky particles (those that pass through the 50-percent openings) increases with increase in specific gravity. Table 10, which gives data for the upper- and lower-bench Acmar coals and similar data for other Alabama coals, gives an idea of the relative importance of the rash problem in the Cahaba field compared with the

Warrior field. The data are given for the 1.38-1.50 and 1.50-1.70 specific-gravity fractions of the minus 3/16-inch plus 14-mesh sizes. The coals from the Cahaba field contain a much higher percentage of rashy material than those from the Warrior field. This difference in the quantity of flaky material partly accounts for the fact that the slack sizes from the Warrior field can be cleaned more efficiently on jigs than can the slack sizes from the Cahaba field.

According to data obtained during the present investigation of coal-washing practice in Alabama, the rash problem is not particularly serious, as jigs and tables (the only type of equipment used in Alabama) can be operated in such a way as to deal quite efficiently with rash. Jigs can remove rash quite effectively, provided they are operated with a strong suction stroke and have rather large size screen-plate perforations. In the case of tables, the removal of rash may be facilitated by using high riffles in conjunction with the minimum amount of wash water consistent with good bed mobility on the deck.

SUMMARY

The problem of washing the Henry Ellen coal at Acmar is one of ash reduction. The two benches that comprise the bed differ greatly in ash content. If provision is made for the rejection of flaky particles known as "rash" it should be possible to wash the top bench to about 13.2 percent ash and the bottom bench to about 5.9 percent ash with good efficiency. Previous work on other coals has indicated that the removal of rash is facilitated when jigs are used by operating with a strong suction stroke and large screen perforations, and when tables are employed by using high riffles and a minimum amount of wash water.

Recrushing of top-bench coal coarser than 3/4-inch prior to washing makes possible additional reduction of ash for the total top bench of only about 0.7 percent.

TABLE 1. - Henry Ellen bed sampled at the Acmar No. 5 mine.
Analyses of standard face samples

Determination	Top Bench			Bottom Bench		
	Coal as received, percent	Coal, moisture-free percent	Coal, moisture-and ash-free, percent	Coal as received, percent	Coal, moisture-free, percent	Coal, moisture-and ash-free, percent
Proximate analysis:						
Moisture.....	2.1	-	-	2.1	-	-
Volatile matter..	29.6	30.2	36.0	32.0	32.7	35.4
Fixed carbon.....	52.5	53.7	64.0	58.4	59.7	64.6
Ash.....	15.8	16.1	-	7.5	7.6	-
Ultimate analysis:						
Hydrogen.....	4.6	4.5	5.4	5.0	4.8	5.2
Carbon.....	69.9	71.4	85.1	77.2	78.9	85.4
Nitrogen.....	1.5	1.6	1.9	1.7	1.8	1.9
Oxygen.....	7.4	5.6	6.7	3.1	6.4	6.9
Sulphur.....	.8	.8	.9	.5	.5	.6
Ash.....	15.8	16.1	-	7.5	7.6	-
Calorific values:						
Calories.....	6,872	7,017	8,367	7,583	7,750	8,389
British thermal units.....	12,370	12,630	15,060	13,650	13,950	15,100
	Softening temperature of ash, 2,610°F.			Softening temperature of ash, 2,390°F.		

TABLE 2. - Henry Ellen bed sampled at Acmar No. 5 mine. Sulphur analyses and ash fusibility. Raw coal through 3/4-inch on 3/16-inch rectangular-mesh screens.

Specific gravity	Softening temperature °F.[1]	Sulphur[2], percent
Float on 1.28.....	2520	0.56
1.28 - 1.30.....	2810	.54
1.30 - 1.38.....	2800	.62
1.38 - 1.50.....	2750	.84
1.50 - 1.70.....	2460	1.47
1.70 - 1.90.....	2510	1.21
1.90 - 2.20.....	2690	.74
Sink in 2.20.....	2590	2.70

1/ The softening temperature is the temperature at which the cone of coal ash is fused down to a spherical lump.

2/ Moisture-free basis.

TABLE 3. - Henry Ellen bed, top bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilograms	Weight, percent	Ash,1/ percent	Cumulative weight, percent	Cumulative ash,1/ percent
Through 4-inch on 1 1/2-inch. Head Ash,1/ percent, not determined. Weight, kilograms, 538.5. Weight, percent, 27.4.	Float on 1.28	2.10	0.6	5.0	0.6	5.0
	1.28 - 1.30	8.20	2.4	7.0	3.0	6.6
	1.30 - 1.38	208.10	60.5	12.1	63.5	11.8
	1.38 - 1.50	85.90	25.0	20.8	88.5	14.4
	1.50 - 1.70	18.90	5.5	33.4	94.0	15.5
	1.70 - 1.90	13.40	3.9	50.2	97.9	16.9
	1.90 - 2.20	5.10	1.5	61.8	99.4	17.5
	Sink in 2.20	2.10	.6	85.9	100.0	18.0
		343.80				
Through 1 1/2-inch on 3/4-inch. Head ash1/ percent, 18.3. Weight, kilograms, 337.2. Weight, percent, 17.2	Float on 1.28	1.40	1.1	5.0	1.1	5.0
	1.28 - 1.30	7.20	5.6	7.1	6.7	6.8
	1.30 - 1.38	76.70	59.4	12.4	66.1	11.8
	1.38 - 1.50	27.60	21.4	20.6	87.5	14.0
	1.50 - 1.70	7.40	5.7	34.4	93.2	15.2
	1.70 - 1.90	4.85	3.7	50.5	96.9	16.6
	1.90 - 2.20	2.62	2.0	62.4	98.9	17.5
	Sink in 2.20	1.42	1.1	83.5	100.0	18.2
		129.19				
Through 3/4-inch on 3/16-inch. Head ash,1/ percent, 16.8. Weight, kilograms, 612.4. Weight, percent, 31.2.	Float on 1.28	1.18	2.0	3.3	2.0	3.3
	1.28 - 1.30	5.54	9.4	5.4	11.4	5.0
	1.30 - 1.38	32.50	55.4	11.1	66.8	10.1
	1.38 - 1.50	11.80	20.1	20.3	86.9	12.4
	1.50 - 1.70	3.72	6.3	32.8	93.2	13.8
	1.70 - 1.90	2.06	3.6	49.4	96.8	15.1
	1.90 - 2.20	1.12	1.9	62.7	98.7	16.0
	Sink in 2.20	.76	1.3	82.5	100.0	16.9
		58.68				

1/ Moisture-free basis.

TABLE 3. - Henry Ellen bed, top bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and-sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilo-grams	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
Through 3/16-inch on 14-mesh. Head ash, 1/ percent, 17.6. Weight, kilograms, 305.9. Weight, percent, 15.6.	Float on 1.28	1.72	9.0	2.3	9.0	2.3
	1.28 - 1.30	2.56	13.4	4.1	22.4	3.4
	1.30 - 1.38	7.82	40.8	9.9	63.2	7.6
	1.38 - 1.50	3.58	18.7	19.6	81.9	10.3
	1.50 - 1.70	1.55	8.1	32.6	90.0	12.3
	1.70 - 1.90	.69	3.6	49.9	93.6	13.8
	1.90 - 2.20	.54	2.8	63.8	96.4	15.2
	Sink in 2.20	.70	3.6	83.7	100.0	17.7
		19.16				
		Grams				
Through 14-mesh on 35-mesh. Head ash, 1/ percent, not determined. Weight, Kilograms, 99.7. Weight, percent, 5.1.	Float on 1.28	20.67	30.8	3.1	30.8	3.1
	1.28 - 1.30	7.41	11.0	7.6	41.8	4.3
	1.30 - 1.38	14.11	21.0	12.6	62.8	7.1
	1.38 - 1.50	11.68	17.4	22.2	80.2	10.3
	1.50 - 1.70	4.93	7.3	36.1	87.5	12.5
	1.70 - 1.90	2.81	4.2	52.1	91.7	14.3
	1.90 - 2.20	1.38	2.0	66.4	93.7	15.4
	Sink in 2.20	4.21	6.3	80.7	100.0	19.5
		67.20				
Through 35-mesh on 100-mesh. Head ash, 1/ percent, not determined. Weight, kilograms, 48.9. Weight, percent, 2.5.	Float on 1.28	10.19	31.0	2.8	31.0	2.8
	1.28 - 1.30	3.28	10.0	7.7	41.0	4.0
	1.30 - 1.38	6.15	18.7	12.2	59.7	6.6
	1.38 - 1.50	5.81	17.7	21.9	77.4	10.1
	1.50 - 1.70	2.72	8.3	35.6	85.7	12.5
	1.70 - 1.90	1.47	4.5	51.2	90.2	14.5
	1.90 - 2.20	.90	2.8	63.2	93.0	15.9
	Sink in 2.20	2.31	7.0	72.1	100.0	19.9
		32.83				

1/ Moisture-free basis.

TABLE 3. - Henry Ellen bed, top bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and-sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilograms	Weight percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
		Grams				
Through 100-mesh.	Float on 1.28	3.35	28.7	2.5	28.7	2.5
Head ash, 1/ percent,	1.28 - 1.30	1.39	10.4	6.2	39.1	3.5
not determined.	1.30 - 1.38	2.42	18.1	11.0	57.2	5.9
Weight, kilograms, 19.9.	1.38 - 1.50	2.50	18.6	20.4	75.8	9.4
Weight, percent, 1.0.	1.50 - 1.70	1.49	11.1	30.8	86.9	12.2
	1.70 - 1.90	.70	5.2	48.4	92.1	14.2
	1.90 - 2.20	.43	3.2	56.4	95.3	15.6
	Sink in 2.20	.63	4.7	64.0	100.0	17.9
		13.41				
Composite, through	Float on 1.28	.....	5.0	3.0	5.0	3.0
4-inch. Head ash, 1/	1.28 - 1.30	.....	7.6	5.6	12.6	4.6
percent, not deter-	1.30 - 1.38	.....	52.1	11.6	64.7	10.2
mined. Weight, kilo-	1.38 - 1.50	.....	21.2	20.5	85.9	12.8
grams, 1,962.5.	1.50 - 1.70	.....	6.4	33.4	92.3	14.2
Weight, percent, 100.0	1.70 - 1.90	.....	3.8	50.1	96.1	15.6
	1.90 - 2.20	.....	2.0	62.8	98.1	16.6
	Sink in 2.20	.....	1.9	81.5	100.0	17.8

1/ Moisture-free basis

TABLE 4. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and-sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilograms	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
Through 4-inch on 1 1/2-inch. Head ash, 1/ ... percent, not determined. Weight, kilograms, 874.9. Weight, percent, 48.3.	Float on 1.28	19.40	6.6	3.6	6.6	3.6
	1.28 - 1.30	174.40	59.1	4.9	65.7	4.8
	1.30 - 1.38	81.90	27.8	8.4	93.5	5.8
	1.38 - 1.50	15.40	5.2	17.0	98.7	6.4
	1.50 - 1.70	2.07	.7	28.0	99.4	6.6
	1.70 - 1.90	.16	.1	32.8	99.5	6.6
	1.90 - 2.20	.12	.1	57.6	99.6	6.7
	Sink in 2.20	1.29	.4	55.4	100.0	6.9
		294.74				
Through 1 1/2-inch on 3/4-inch. Head ash, 1/ percent, 6.9. Weight, kilograms, 298.4. Weight, percent, 16.5.	Float on 1.28	17.00	11.7	3.5	11.7	3.5
	1.28 - 1.30	76.50	52.4	4.9	64.1	4.6
	1.30 - 1.38	42.70	29.3	8.1	93.4	5.7
	1.38 - 1.50	6.10	4.2	18.4	97.6	6.3
	1.50 - 1.70	2.24	1.5	26.6	99.1	6.6
	1.70 - 1.90	.57	.4	38.5	99.5	6.7
	1.90 - 2.20	.34	.2	46.8	99.7	6.8
	Sink in 2.20	.49	.3	68.6	100.0	7.0
		145.94				
Through 3/4-inch on 3/16-inch. Head ash, 1/ percent, 7.2. Weight, kilograms, 388.0. Weight, percent, 21.4.	Float on 1.28	10.00	17.1	3.3	17.1	3.3
	1.28 - 1.30	24.80	42.4	4.6	59.5	4.2
	1.30 - 1.38	19.00	32.5	8.0	92.0	5.6
	1.38 - 1.50	2.70	4.6	17.7	96.6	6.1
	1.50 - 1.70	1.28	2.2	28.1	98.8	6.6
	1.70 - 1.90	.31	.5	40.4	99.3	6.8
	1.90 - 2.20	.16	.3	54.1	99.6	6.9
	Sink in 2.20	.20	.4	72.6	100.0	7.2
		58.45				

1/ Moisture-free basis

TABLE 4. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and-sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilograms	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
Through 3/16-inch on 14-mesh. Head ash, 1/ percent, 9.3. Weight, kilograms, 164.3. Weight, percent, 9.1.	Float on 1.28	2.58	17.6	2.3	17.6	2.3
	1.28 - 1.30	5.57	38.0	4.0	55.6	3.5
	1.30 - 1.38	4.32	29.5	8.3	85.1	5.1
	1.38 - 1.50	1.07	7.3	17.6	92.4	6.1
	1.50 - 1.70	.56	3.8	31.6	96.2	7.1
	1.70 - 1.90	.23	1.6	47.5	97.8	7.8
	1.90 - 2.20	.14	1.0	60.8	98.8	8.3
	Sink in 2.20	.17	1.2	80.3	100.0	9.2
		14.64				
		Grams				
Through 14-mesh on 35-mesh. Head ash, 1/ percent, not determined. Weight, kilograms, 52.5. Weight, percent, 2.9.	Float on 1.28	30.16	56.4	2.9	56.4	2.9
	1.28 - 1.30	6.22	11.6	7.8	68.0	3.7
	1.30 - 1.38	7.41	13.9	13.0	81.9	5.3
	1.38 - 1.50	3.70	6.9	22.5	88.8	6.6
	1.50 - 1.70	2.63	4.9	37.9	93.7	8.3
	1.70 - 1.90	1.30	2.4	54.2	96.1	9.4
	1.90 - 2.20	.71	1.3	66.4	97.4	10.2
	Sink in 2.20	1.40	2.6	79.4	100.0	12.0
		53.53				
Through 35-mesh on 100-mesh. Head ash, 1/ percent, not determined. Weight, kilograms, 23.6. Weight, percent, 1.3.	Float on 1.28	11.65	48.7	2.5	48.7	2.5
	1.28 - 1.30	2.66	11.1	7.3	59.8	3.4
	1.30 - 1.38	3.65	15.3	12.6	75.1	5.3
	1.38 - 1.50	2.02	8.4	22.6	83.5	7.0
	1.50 - 1.70	1.49	6.2	37.4	89.7	9.1
	1.70 - 1.90	.84	3.5	53.7	93.2	10.8
	1.90 - 2.20	.55	2.3	65.2	95.5	12.1
		23.94				

1/ Moisture-free basis.

TABLE 4. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Screen-sizing and float-and-sink data of raw coal, crushed to pass a 4-inch round-hole screen.

Description	Specific gravity	Weight, kilo-grams	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
		Grams				
Through 100-mesh.	Float on 1.28	3.66	38.8	2.2	38.8	2.2
Head ash, 1/ percent,	1.28 - 1.30	1.09	11.6	6.3	50.4	3.1
not determined.	1.30 - 1.38	1.62	17.2	11.6	67.6	5.3
Weight, kilograms, 9.1.	1.38 - 1.50	1.04	11.0	20.4	78.6	7.4
Weight, percent, 0.5.	1.50 - 1.70	.93	9.9	31.8	88.5	10.1
	1.70 - 1.90	.39	4.1	51.1	92.6	12.0
	1.90 - 2.20	.19	2.0	61.1	94.6	13.0
	Sink in 2.20	.51	5.4	74.5	100.0	16.3
		9.43				
Composite, through 4-inch. Head ash, 1/ percent, not determined. Weight, kilograms, 1811.3. Weight, percent, 100.0	Float on 1.28	.....	12.8	3.2	12.8	3.2
	1.28 - 1.30	.....	50.3	4.8	63.1	4.5
	1.30 - 1.38	.....	28.6	8.4	91.7	5.7
	1.38 - 1.50	.....	5.2	17.8	96.9	6.3
	1.50 - 1.70	.....	1.7	30.0	98.6	6.8
	1.70 - 1.90	.....	0.5	45.0	99.1	6.9
	1.90 - 2.20	.....	0.3	58.6	99.4	7.1
	Sink in 2.20	.....	0.6	69.3	100.0	7.5

1/ Moisture-free basis

TABLE 5. - Henry Ellen bed sampled at Acmar No. 5 mine. Float-and-sink data of composite top and bottom bench raw coal crushed to pass a 4-inch round-hole screen

Description	Specific gravity	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash, 1/ percent
Total bed.	Float on 1.28	8.5	3.1	8.5	3.1
Through 4-inch.	1.28 - 1.30	26.7	4.9	35.2	4.5
Head ash, 1/ percent,	1.30 - 1.38	41.6	10.6	76.8	7.8
not determined.	1.38 - 1.50	14.1	20.1	90.9	9.7
	1.50 - 1.70	4.3	32.8	95.2	10.7
	1.70 - 1.90	2.3	49.6	97.5	11.7
	1.90 - 2.20	1.2	62.3	98.7	12.3
	Sink in 2.20	1.3	79.0	100.0	13.1

TABLE 6. - Summary of washability data. Values taken at points on the washability curves corresponding to readings of 10.0 percent and 15.0 percent on the ± 0.10 specific-gravity distribution curves

Screen size	10 percent ± 0.10			15 percent ± 0.10		
	Specific gravity	Yield, percent	Ash, percent	Specific gravity	Yield, percent	Ash, percent
		Top bench				
Composite, 4- to 0-inch	1.543	87.8	13.2	1.513	86.5	12.9
4- to 1 1/2-inch	1.538	90.2	14.6	1.518	89.4	14.5
1 1/2- to 3/4 inch	1.532	88.8	14.2	1.508	87.9	14.1
3/4- to 3/16-inch	1.531	88.2	12.6	1.505	87.1	12.4
3/16-inch to 14-mesh	1.576	86.0	11.2	1.523	83.5	10.6
14- to 35-mesh	1.571	84.0	11.3	1.530	82.2	10.8
35- to 100-mesh	1.585	82.0	11.3	1.532	79.5	10.6
Through 100-mesh	1.630	84.0	11.4	1.565	81.0	10.6
		Bottom bench				
Composite, 4- to 0-inch	1.442	95.0	5.9	1.427	94.3	5.8
4- to 1 1/2-inch	1.443	97.3	6.0	1.424	96.5	6.0
1 1/2- to 3/4-inch	1.423	95.2	5.0	1.414	95.0	5.9
3/4- to 3/16-inch	1.435	94.6	5.8	1.422	94.1	5.8
3/16-inch to 14-mesh	1.474	91.4	5.9	1.434	89.2	5.6
14- to 35-mesh	1.476	87.9	6.4	1.428	85.5	5.9
35- to 100-mesh	1.515	84.3	7.2	1.455	81.0	6.4
Through 100-mesh	1.610	35.3	9.0	1.515	79.7	7.7
		Total bed				
Through 4-inch	1.526	91.7	9.8	1.488	90.0	9.6

TABLE 7. - Henry Ellen bed, top bench, sampled at Acmar No. 5 mine. Float-and-sink data of raw coal samples crushed to pass a 3/16-inch square-mesh screen.

Description	Specific gravity	Weight, percent	Ash, 1/ percent	Cumulative weight, percent	Cumulative ash 1/ percent
Through 4-inch on 1 1/2-inch, crushed to pass 3/16-inch. Composite through 3/16-inch.	Float on 1.28	15.4	3.4	15.4	3.4
	1.28 - 1.30	9.8	6.5	25.2	4.6
	1.30 - 1.38	37.0	11.8	62.2	8.9
	1.38 - 1.50	19.8	21.0	82.0	11.8
	1.50 - 1.70	9.0	33.8	91.0	14.0
	1.70 - 1.90	5.0	52.2	96.0	16.0
	1.90 - 2.20	2.8	63.4	98.8	17.3
	Sink in 2.20	1.2	80.0	100.0	18.1
Through 1 1/2-inch on 3/4-inch, crushed to pass 3/16-inch. Composite through 3/16-inch.	Float on 1.28	12.6	3.3	12.6	3.3
	1.28 - 1.30	8.8	6.0	21.4	4.4
	1.30 - 1.38	39.2	11.4	60.6	8.9
	1.38 - 1.50	21.8	20.9	82.4	12.1
	1.50 - 1.70	9.3	34.2	91.7	14.3
	1.70 - 1.90	4.6	51.4	96.3	16.1
	1.90 - 2.20	2.2	62.5	98.5	17.1
	Sink in 2.20	1.5	83.0	100.0	18.1

1/ Moisture-free basis

TABLE 8. - Henry Ellen bed, top bench sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings,[1] percent		Specific-gravity product, 1.38 to 1.50		Specific-gravity product, 1.50 to 1.70		Specific-gravity product, 1.70 to 1.90	
	Through	On	Weight[2] percent	Total weight[2] percent	Weight,[2] percent	Total weight,[2] percent	Weight[2] percent	Total weight[2] percent
Through 3/4-inch on 0.525-inch.	-	100	23.6		17.1		14.8	
	100	75	31.2		28.5		26.1	
	75	50	36.1		32.1		35.2	
	50	-	9.1	27.8	22.3	27.7	23.9	25.9
			100.0		100.0		100.0	
Through 0.525-inch on 0.371-inch.	-	100	11.8		7.4		11.3	
	100	75	35.4		28.0		23.2	
	75	50	41.6		44.3		43.4	
	50	-	11.2	29.4	20.3	25.6	22.1	25.1
			100.0		100.0		100.0	
Through 0.371-inch on 3-mesh	-	100	20.8		13.0		12.0	
	100	75	37.9		30.6		28.0	
	75	50	31.7		34.0		37.6	
	50	-	9.6	22.3	22.4	21.9	22.4	24.5
			100.0		100.0		100.0	
Through 3-mesh on 4- mesh.	-	100	14.8		11.1		8.4	
	100	75	39.2		29.6		26.3	
	75	50	25.8		30.2		25.7	
	50	-	20.2	20.5	29.1	24.8	39.6	24.5
			100.0	100.0	100.0	100.0	100.0	100.0

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.
2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 8. - Henry Ellen bed, top bench sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings[1], percent: Through	On	Specific-gravity product, 1.38 to 1.50: Weight[2], percent	Total weight[2], percent	Specific-gravity product, 1.50 to 1.70: Weight[2], percent	Total weight[2], percent	Specific-gravity product, 1.70 to 1.90: Weight[2], percent	Total weight[2], percent
Composite through 3/4-inch on 4-mesh.	-	100	17.7		12.2		11.7	
	100	75	35.6		29.1		25.9	
	75	50	34.6		35.2		35.5	
	50	-	12.1		23.5		26.9	
			100.0		100.0		100.0	
Through 4-mesh on 6-mesh.	-	100	15.8		9.7		8.3	
	100	75	33.5		23.2		18.9	
	75	50	38.4		42.7		41.9	
	50	-	12.3	34.8	24.4	36.5	30.9	31.9
			100.0		100.0		100.0	
Through 6-mesh on 8-mesh.	-	100	10.7		8.8		8.9	
	100	75	52.3		28.0		31.0	
	75	50	27.2		37.2		35.1	
	50	-	9.8	22.5	26.0	21.9	25.0	24.7
			100.0		100.0		100.0	
Through 8-mesh on 10-mesh.	-	100	15.6		10.0		11.3	
	100	75	37.0		22.1		23.9	
	75	50	41.6		51.4		47.9	
	50	-	5.8	22.1	16.5	18.7	16.9	20.8
			100.0		100.0		100.0	
Through 10-mesh on 14-mesh.	-	100	9.8		6.8		7.1	
	100	75	44.9		39.4		24.7	
	75	50	31.8		34.9		35.7	
	50	-	13.5	20.6	18.9	22.9	32.5	22.6
			100.0	100.0	100.0	100.0	100.0	

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.

2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 8. - Henry Ellen bed, top bench sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings[1], percent		Specific-gravity product, 1.38 to 1.50		Specific-gravity product, 1.50 to 1.70		Specific-gravity product, 1.70 to 1.90	
	Through	On	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent
Composite through 4-mesh on 14-mesh.	-	100	13.4		8.9		8.8	
	100	75	40.9		27.8		24.2	
	75	50	35.2		41.3		40.1	
	50	-	10.5		22.0		26.9	
			100.0		100.0		100.0	

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.
2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 9. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings[1], percent		Specific gravity product, 1.38 to 1.50		Specific gravity product 1.50 to 1.70		Specific gravity product, 1.70 to 1.90	
	Through	On	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent
Through 3/4-inch on 1/2-inch	-	100	20.7		16.9		0.0	
	100	75	26.4		32.8		9.1	
	75	50	41.3		41.7		48.5	
	50	-	11.6	31.0	8.6	26.4	42.4	21.1
			100.0		100.0		100.0	
Through 1/2-inch on 0.371-inch	-	100	9.7		10.3		3.3	
	100	75	27.9		27.0		19.6	
	75	50	43.9		43.0		46.7	
	50	-	18.5	24.4	19.7	29.8	30.4	29.5
			100.0		100.0		100.0	
Through 0.371-inch on 3-mesh	-	100	14.7		15.0		4.4	
	100	75	33.4		29.4		27.9	
	75	50	31.6		31.5		29.4	
	50	-	20.3	22.3	24.1	22.4	38.3	21.8
			100.0		100.0		100.0	
Through 3-mesh on 4-mesh	-	100	10.9		10.9		9.3	
	100	75	27.6		23.4		24.4	
	75	50	28.9		23.4		25.6	
	50	-	32.6	22.3	42.3	21.4	40.7	27.6
			100.0	100.0	100.0	100.0	100.0	100.0

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.
2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 9. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings[1], percent: Through	On	Specific-gravity product, 1.38 to 1.50: Weight[2] percent	Total weight[2] percent	Specific-gravity product, 1.50 to 1.70: Weight[2] percent	Total weight[2] percent	Specific-gravity product, 1.70 to 1.90: Weight[2] percent	Total weight[2] percent
Composite through 3/4-inch on 4-mesh	-	100	14.5		13.2		4.5	
	100	75	28.6		28.3		20.5	
	75	50	37.0		35.9		37.5	
	50	-	19.9		22.6		37.5	
			100.0		100.0		100.0	
Through 4-mesh on 6-mesh	-	100	12.0		7.6		5.3	
	100	75	23.3		15.8		18.7	
	75	50	40.8		31.5		30.7	
	50	-	23.9	30.3	45.1	32.9	45.3	31.9
			100.0		100.0		100.0	
Through 6-mesh on 8-mesh	-	100	9.5		5.0		7.0	
	100	75	33.9		25.2		24.6	
	75	50	33.9		28.6		22.8	
	50	-	22.7	20.5	41.2	21.3	45.6	24.2
			100.0		100.0		100.0	
Through 8-mesh on 10-mesh	-	100	13.1		7.1		8.2	
	100	75	26.9		15.8		16.3	
	75	50	50.5		47.2		40.8	
	50	-	9.5	26.3	29.9	22.7	34.7	20.9
			100.0		100.0		100.0	
Through 10-mesh on 14-mesh	-	100	6.9		5.4		7.4	
	100	75	39.0		23.3		24.1	
	75	50	33.7		31.8		24.1	
	50	-	20.4	22.9	39.5	23.1	44.4	23.0
			100.0	100.0	100.0	100.0	100.0	100.0

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.
2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 9. - Henry Ellen bed, bottom bench, sampled at Acmar No. 5 mine. Rectangular-mesh screen tests of raw coal through 3/4-inch on 3/16-inch, and through 3/16-inch on 14-mesh.

Products	Ratio of rectangular mesh openings to minimum square-mesh openings[1], percent		Specific-gravity product, 1.38 to 1.50		Specific-gravity product, 1.50 to 1.70		Specific-gravity product, 1.70 to 1.90	
	Through	On	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent	Weight[2] percent	Total weight[2] percent
Composite through 4-mesh on 14-mesh	-	100	10.6		6.5		6.8	
	100	75	30.0		19.5		20.8	
	75	50	40.3		34.5		29.4	
	50	-	19.1		39.5		43.0	
			100.0		100.0		100.0	

1/ Width of rectangular mesh divided by opening of square mesh on which the product is retained, in percent.
2/ All products were dried on a steam drier before weighing. The residual moisture in all products after drying was approximately 1.0 percent.

TABLE 10. - Comparative flakiness data on coals from various mines in Warrior and Cahaba fields

Specific gravity	Rectangular mesh screens, percent opening		Warrior field					
			Mary Lee bed at-			Black Creek bed at Bradford	Blue Creek bed at Connellsville	Brookwood bed at Warrior View
			Lewisburg	Sayreton	Hull			
	Through	On	Weight[1] percent	Weight[1] percent	Weight[1] percent	Weight[1] percent	Weight[1] percent	Weight[1] percent
1.38 to 1.50	-	75	5.2	8.7	0.3	2.0	13.9	5.3
	75	50	2.4	2.4	3.2	.8	1.9	1.6
	50	-	.9	.4	.8	.2	.3	.7
1.50 to 1.70	-	75	2.2	2.9	2.4	1.1	7.7	2.3
	75	50	2.1	1.2	1.2	.6	1.6	1.1
	50	-	.8	.3	.5	.2	.2	.5

1/ Weight percentages in terms of raw coal through 3/16-inch on 14-mesh.

Specific gravity	Rectangular mesh screens, percent opening		Cahaba field				
			Clark bed at Marvel	Woodstock bed at Hills Creek	Thompson bed at Coleanor	Henry Ellen bed, at Acmar, top bench	Bottom bench
	Through	On	Weight[1] percent	Weight[1] percent	Weight[1] percent	Weight[1] percent	Weight[1] percent
1.38 to 1.50	-	75	5.0	5.0	6.3	10.2	3.1
	75	50	4.1	4.1	4.1	6.6	2.7
	50	-	1.9	1.8	4.0	2.0	1.5
1.50 to 1.70	-	75	2.8	2.8	2.1	3.0	1.6
	75	50	3.1	3.2	2.1	3.3	1.4
	50	-	1.5	1.6	1.1	1.8	.9

1/ Weight percentages in terms of raw coal through 3/16-inch on 14-mesh.

AFTER THIS REPORT HAS SERVED YOUR PURPOSE AND IF YOU HAVE NO FURTHER NEED FOR IT, PLEASE RETURN IT TO THE BUREAU OF MINES. THE USE OF THIS MAILING LABEL TO DO SO WILL BE OFFICIAL BUSINESS AND NO POSTAGE STAMPS WILL BE REQUIRED.

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

OFFICIAL BUSINESS

RETURN PENALTY LABEL

THIS LABEL MAY BE USED ONLY FOR RETURNING OFFICIAL PUBLICATIONS. THE ADDRESS MUST NOT BE CHANGED

PENALTY FOR PRIVATE USE TO AVOID PAYMENT OF POSTAGE, $300

BUREAU OF MINES,

WASHINGTON, D. C.

UNIVERSITY OF ILLINOIS-URBANA
3 0112 055265638